SEPTEMBER 1952

MINING ENGINEERING

MILL DESIGN TAILORED FOR CUSTOM ORES . . .

MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 4 NO. 9 SEPTEMBER, 1952

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FEATURES

ARTICLES

TRANSACTIONS

Letters to the Editor

The Broadening Road to Foreign Investment

I do not think too much emphasis can be placed upon creation of favorable climate for development of foreign mineral resources to feed the hungry maw of our industrialized nation. We have become dependent, not only upon the ten raw products indicated in Mr. Bancroft's article, but upon 71 mineral substances listed by DMPA as in short supply. Forty-four of these are metals, 9 are chemical minerals and 18 are nonmetallic minerals. Of this list only 8 are produced in any large quantity in the United States, and all are vital in peace-time industry. Even more, they are absolutely essential for defense.

This represents our present needs. But if our consumption and requirements are projected over the next 25 years as is done by the President's Materials Policy Commission—The Paley Report—our future outlook becomes appalling. The divergence of consumption over production, even allowing for substitution and conservation, will widen in the future. This means our dependence upon foreign raw materials, mainly minerals, will grow.

What are the chief foreign sources of minerals for our future needs? They are chiefly Latin America and the dominions and colonies of the Western European nations. The British Commonwealth countries contain in abundance those minerals which the U. S. lacks. Of 32 of the most critical minerals the Commonwealth is deficient in 7 and lacks only 1; whereas the U. S. is deficient in 18. Strikingly, our deficient minerals are abundant in the Commonwealth and several of their deficiencies are in excess in the U. S. Resources of the two nations complement each other.

If Europe should be overrun by an unfriendly power that could control the destinies and resources of the European dominions and colonies, our industrial life would be placed in severe jeopardy. Therefore, I heartily endorse Mr. Bancroft's statements regarding the Point IV program, provided the program is wisely administered and implemented. It is from the underdeveloped countries, as pointed out by Mr. Bancroft, that most of our future mineral supplies will have to come.

Many of the underdeveloped source countries are African colonies and dominions of Western European nations. Our present aid to Europe, therefore, is not, as many infer, a purely one-sided affair; it safeguards our future industrial development and security.

Development aid directly to the underdeveloped countries through a well administered Point IV program, can do much, I believe, to safeguard our prosperity. I regard it as a wise investment for future U. S. mineral needs.

Dr. Alan M. Bateman
Department of Geological Sciences
Yale University

Mr. Howland Bancroft's article, The Broadening Road to Foreign Investment, appeared on page 666 of the July MINING ENGINEERING.

Spirals Recover Heavy Mineral By-product

W. R. Hudspeth in his article on page 767 August MINING ENGINEERING tells us that: "screen oversize . . . is pumped to a hydrocyclone for desliming. . ." What becomes of the undersize at this point?

Christian S. Anderson
New York

The text in full should have read: The screen oversize is returned to the heads of the mills by means of an Esperanza drag, the screen undersize is pumped to a 12-in. hydrocyclone for desliming.

Ready Anytime
Eimco Loaders are ready to go to work anytime of the day or night. These heavy-duty units are built of strong alloy-steel castings, which are carefully machined and assembled. They are built to stay on the job, no matter how difficult or long it may be.
Write for information on jobs Eimco has done which may be similar to yours.
EIMCO
THE EIMCO CORPORATION
The World's Largest Manufacturers of Underground Rock Loading Machines
BRANCH SALES AND SERVICE OFFICES

—Personnel Service—

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1**. Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

POSITIONS OPEN

Mining Engineer with some engineering training, capable of doing mapping, surveying, exploration, geology and transit work. Location, New York State. Y7333.

Mineral Dressing Engineer, B.S. Mining or Chemical Engineering, age to 35. Must have knowledge of flotation. Duties will be in applied research and field evaluation of cationic and anionic flotation agents. Salary, $3500 to $7000 depending on experience. Location, Chicago. R-8971.

Geological Draftsman, 25 to 30, with some experience in working with a geologist. Salary, $5100 a year. Location, New York, N. Y. Y7153.

WANTED

RESEARCH METALLURGIST

Not over 45 years old with excellent technical and practical experience in flotation, concentration, copper leaching, for long established large East Mediterranean copper pyrite mill. We offer three-year contract; single status 6 months; 3 months' paid vacation at contract end. Free housing; California climate; pension plan non-contributory. All travel expenses self and family; children not over 12 years old. Must pass life insurance physical. References required, also photo, complete personal history and minimum acceptable salary. Write Box F-13, MINING ENGINEERING.

ATOMIC ENERGY COMMISSION

Metallurgical engineers with professional training and operating experience in ore processing. Positions involve administration of uranium production programs. Salary range $7,000-$10,000. Positions located in Washington, D. C. and in the field. Address inquiries to George M. Gableman, Chief, Personnel Operations, 1901 Constitution Avenue, N. W., Washington 25, D. C. and enclose a background statement.

Engineers. (a) Graduate Assistants in mining engineering, recent graduates. Salaries, $1120 for ten months with all fees, except the health fee, paid by the college. For this appointment, service is half and program is two-thirds time. By going full-time during the summer it is possible to obtain an M.S. degree in one year. Position starts September 16, 1952. (b) Full time research appointment also available. Salary open depending on experience record. Location, Pennsylvania. Y7326.

Mining Engineer, 25 to 45, with three or more years' experience, to work as shift boss. Excellent opportunity to advance in underground department. Salary, $4650 a year. Location, Canada. Y7243.

Research and Development Engineer to lead program directed toward concentration and recovery of minerals from nonferrous ores. The existing facilities are principally concerned with electro-static and electro-magnetic separations and milling and classifying operations. Should have ability to plan and execute a program with the above objectives in mind. Should have sound technical training, with a few years' experience in research and development work in this field, and preferably some production experience. Location, Pennsylvania. Y7091.

POSITION WANTED

METALLURGICAL ENGINEER, Ph.D. Several years' applied experience and teaching. Extensive knowledge of physical and production metallurgy and associated fields. Desires a good academic position.

Box F-14 MINING ENGINEERING

DO YOU REQUIRE A MANAGER OR DIRECTOR

of

Operations and Explorations

Excellent Record and References (Employed)

Box E-12 MINING ENGINEERING

Research Engineer, 25 to 40, graduate in mining or metallurgical engineering, preferably with a Master's degree in mineral dressing, with at least three years' experience in ore dressing, research or mill work, preferably with experience in nonmetallic flotation. Location, Florida. Y6465.

Assistant Chief Engineer, about 40, with experience in mining and/or chemical plant operations. Should have experience with shovels, hydraulic grinding and classifying. Salary open. Location, Florida. Y7109.

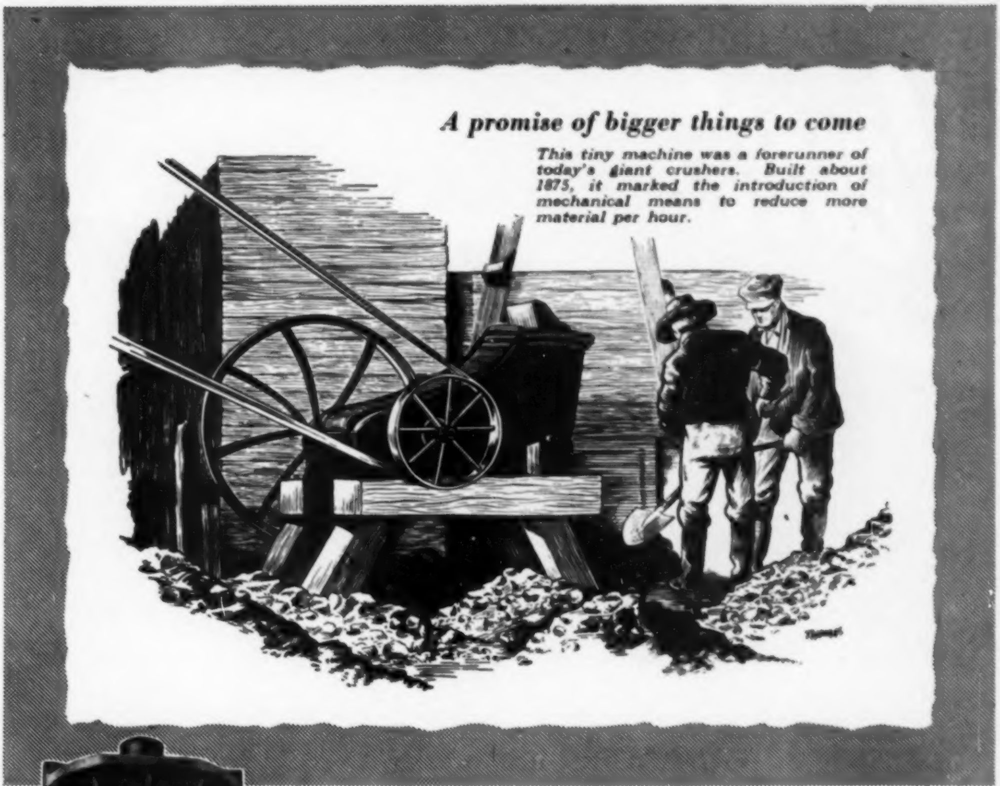
Engineers. (a) Civil or Mining Engineer, 30 to 40, graduate, with some experience in road building and construction for field work. Knowledge of Spanish essential. Location, Caribbean Area. (b) Junior Field Engineer, graduate civil or mining, young, single status, with some experience in road building and construction for mining camp. Spanish desirable. Salary open. Location, Caribbean Area. Y6000.

MEN AVAILABLE

Mine Manager, B.S.E.M., 52, good health. Speak Spanish. Executive positions 22 years. Excellent record low cost mining and labor relations. Experience Mesabi Range, Western Copper and Gold mines. Past 15 years manager of copper, tin and gold dredging organizations in South America. Available after September. M-698.

SALES ENGINEER. Salaried position for engineer experienced in ore dressing or coal preparation. Permanent position with leading equipment manufacturing and engineering construction firm. Submit details of experience, age, salary required and starting date. All replies strictly confidential.

Box F-15 MINING ENGINEERING



The Traylor TC Gyratory, with its curved concaves and bell head, typifies advanced Traylor design for efficient operation. See Bulletin 138 for complete details.

Step by step, little machines and crude inventions have been developed into powerful, more efficient equipment to keep pace with the needs of a growing nation. For 50 years, Traylor has made it a policy to *lead* in the development and production of better machinery for the mining industry. In that time, mining men have come to depend on the skill and experience of Traylor to supply them with the tools they need. They know that when they consult Traylor, they consult experience . . . half a century of it.

OUR GOLDEN ANNIVERSARY YEAR

1902 1952

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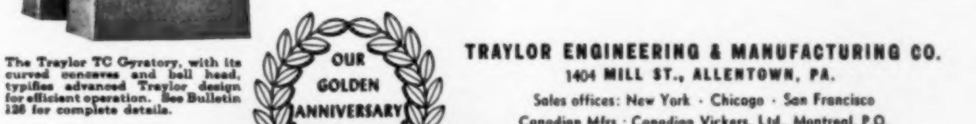
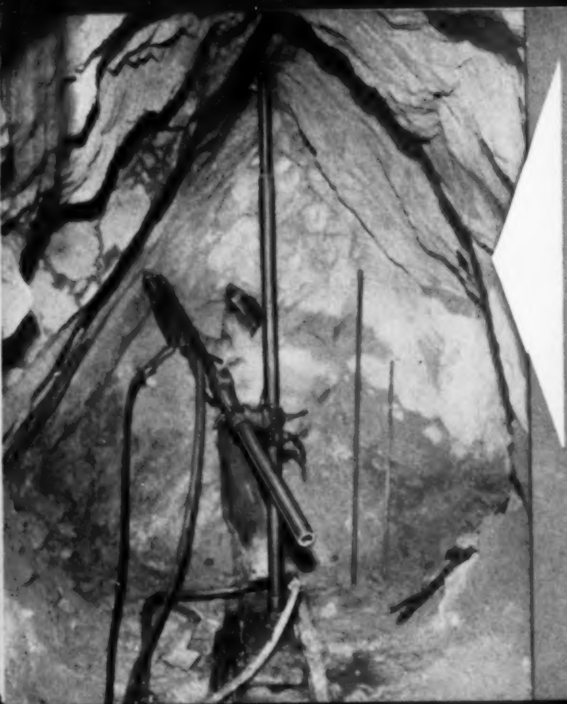
Sales offices: New York · Chicago · San Francisco
Canadian Mfrs.: Canadian Vickers, Ltd., Montreal, P.Q.

a **Traylor** *leads to greater profits*





Miners like Le Roi-CLEVELAND HC23RW Reverse Air Feed Drifters

Management does, too

Faster Steel Changes! No swing or dump nuts to loosen and reset. Your miners simply swing drifter on feed cylinder and change steels. It's not only easy — it lets them drill out the round faster.

No Stuck Steels! Positive air feed keeps drills working at peak efficiency, avoids stuck steels.

Higher Drilling Speeds! Positive air feed plus proper force of blow and strong rotation give faster drilling speeds with both steel and tungsten carbide bits. You get longer bit life, too, and drill more footage.

Low Upkeep Cost! No feed screws or feed-screw nuts to wear. No complicated power-feed mechanism to give trouble.

Easy to Operate! Built to lighten the load on your miners. Feed controls conveniently located. Reverse air feed withdraws steel from hole quickly.

Faster Set-ups! The combination of Le Roi-CLEVELAND Air Feed Drifters and air columns gives you a unit that can be set up easily and quickly. And you can get the air column in any height you want.

drilling cycles

Le Roi-CLEVELAND *self-leveling* Mine Jumbo with four-foot steel-change Air Feed Drifter

Saves time drilling lifters! Lets your miners drill the right round for any ground!

You couldn't ask for more from a mine jumbo than the performance you get from this new Le Roi-CLEVELAND. It's got plenty of stuff. And the payoff for you is faster cycles, greater tonnage per man-shift, lower costs! Here's why:

Self-leveling, air-motor-powered arm, lets miners spot and space holes quickly and easily, for the most efficient fragmentation. They don't have to loosen a bolt or tilt a boom, to complete the drilling cycle.

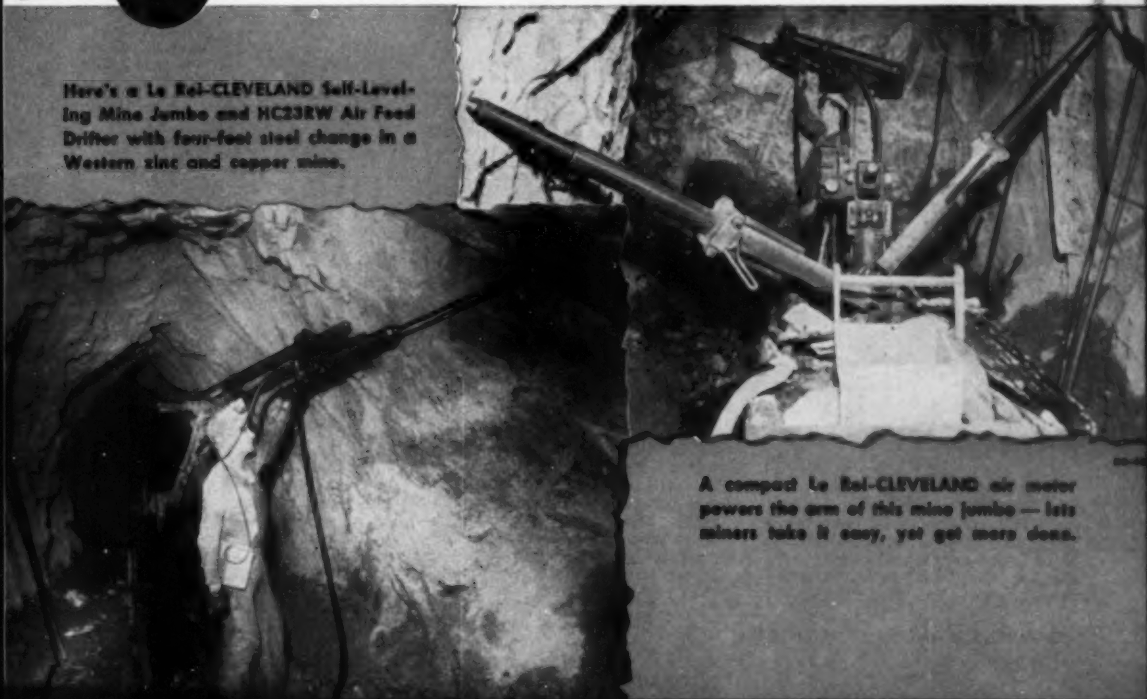
Exclusive rigid screw and gearing mechanism keeps the heading straight, cuts down overbreak and underbreak. Keeps the drifters in line, prevents the steel from binding, reduces chuck wear.

Offset arm provides plenty of clearance to drill lifters — without having to take time out to swing the drill under the arm.

You can get this Le Roi-CLEVELAND Self-Leveling Mine Jumbo in either single-arm or double-arm construction. Write for further information and see for yourself how either model can help you get more done every shift.

LE ROI COMPANY

CLEVELAND ROCK DRILL DIVISION
12500 Berea Road, Cleveland 11, Ohio
Plants: Milwaukee, Cleveland and Greenwich, O.



Here's a Le Roi-CLEVELAND Self-Leveling Mine Jumbo and HC23RW Air Feed Drifter with four-foot steel change in a Western zinc and copper mine.

A compact Le Roi-CLEVELAND air motor powers the arm of this mine jumbo — lets miners take it easy, yet get more done.

Rougher Cells

The photo shows one of three carloads of new *Denver Equipment Co.* rougher flotation machines destined for a single customer.

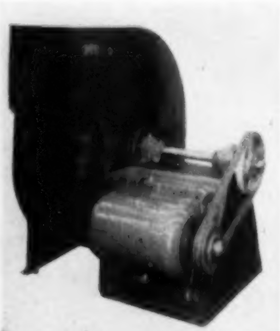
Low level froth overflow on both sides, and double impeller mechanisms in an open-flow type tank, were designed to enable control of variable conditions in the rougher circuit for better metallurgy.

Intense agitation and aeration are claimed, for effective recovery of low-grade mineral particles.

Standard parts are used, interchangeable with regular Denver Sub-A units. The Super Rougher is now in production and a 2-cell No. 24 machine will be displayed at the Mining Show. **Circle No. 1**

Fans

A line of industrial fans built for air or materials handling is available in eleven sizes from 670 to



44,000 cfm capacity from *Westinghouse Electric Corp.* The air handling wheel has backwardly-inclined blades. **Circle No. 2**

Trolleyphones

The Femco Trolleyphone, plug-in model, 2550 series, is offered as a simple mine or mill communications system by *Farmers Engineering & Mfg. Co.* More compact, easier to install and service, the new model is powered from existing trolley wire or power line. **Circle No. 3**

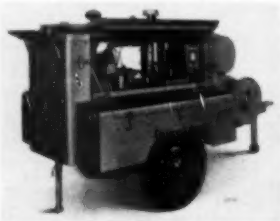
Plastic Pipe

Carlon Products Corp. is producing improved type L rigid plastic pipe, and a new compression type coupling. This pipe has increased burst resistance, higher strength, and greater life expectancy. Being rigid it is suitable for suction lines. A nonconductor, the pipe may also be used for underwater, underground, and concrete imbedded electrical conduit.

New insert ell and tee couplings facilitate sharp turns or take-offs from flexible plastic lines. One fitting now serves for three or four used previously. **Circle No. 4**

Compressors

Ingersoll-Rand rounds out its line of Gyro-Flo portable compressors with 310, 210, and 105 cfm units in addition to its 600 cfm model. Rotary



sliding-vane design is claimed by the manufacturer to give simplicity, low operating cost, and greater reliability. **Circle No. 5**

Blind Bolts

Blind lockbolts made by *Huck Mfg. Co.* are finding use in heavy vehicle maintenance, and other work. Maximum material thickness fastened by this heat treated alloy rivet is 1.594 in. **Circle No. 6**

Conveyors

Link-Belt Co's. exhibit at the 1952 Metal Mining Show will emphasize importance of conveyors for ore haulage. Conveyor idlers and an operating double-deck 5x14 ft CA vibrating screen will be shown, together with pictures of outstanding recent installations. **Circle No. 7**

Colorimeter

A new *Beckman* flow colorimeter simplifies continuous analysis and control of flowing or static liquid or gaseous streams. The measuring phototube compares a standard beam with one passing through the sample. Difference of phototube response yields direct readings or operates control circuits. The unit is designed for a spectral range from 350 to 1000 millimicrons. **Circle No. 8**

Bin Indicator

Bin-Vue is said to be a low-priced, yet fool-proof and accurate bin level indicator, because of simplicity of design. *Convair Co.* states there are no diaphragms or intricate parts. Electrical connections are suitable for signaling, or automatic control of electrical equipment. **Circle No. 9**

Boom

A 600-lb capacity *Yale & Town* electric crane equipped with articulated platform is used for faster and

easier maintenance of lighting and other overhead equipment indoors and out, at the new Caterpillar plant, Joliet, Ill. **Circle No. 10**

Level Control

Photoswitch Inc. liquid level control type 10CB1 features liquid contact only through stainless steel probes. It uses no vacuum tubes, and requires no floats or other moving parts in the tank. **Circle No. 11**

Rubber

Linatex, a special form of natural rubber from Malaya is now supplied in the U. S. by the *Linatex Corp.* in sheet form from 1/32 to 1/2 in. thickness. Workmen on the job can apply Linatex to metal, wood, and concrete surfaces using cold air cured cement. Patented process preserves crystal structure and imparts resistance to low temperature, abrasion, and moisture. **Circle No. 12**

Anchor Bolts

Single and double anchor bolt assemblies developed by *Super-Grip Anchor Bolt Co.* are suitable for masonry, brick, or wood. The bolt can be set at any angle and later bent without disturbing the anchorage. **Circle No. 13**

Copys

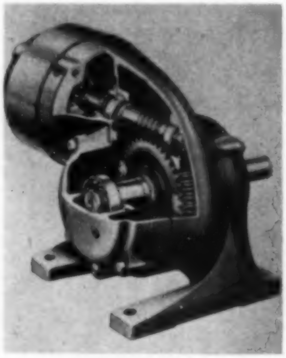
The Copyflex Model 14 by *Charles Bruning Co.* is said to be a tool for reducing paperwork costs. The deskside machine has 20 in. copying width to handle accounting and statistical sheets, or takes letter-size forms two at a time. Average overall cost is stated as less than 2¢ per sq ft of copy. **Circle No. 14**

Free Literature

TEN FEET TALL and four feet wide, tire designed for LeTourneau equipment is world's largest, and will operate with only 10 to 15 pounds air pressure.

(15) SOIL SAMPLING: A complete collection of data about soil sampling techniques, accumulated during the past 33 years by the *Acker Drill Co.*, is displayed in a 16-page bulletin. Modern sampling techniques are discussed along with recommendations as to correct tools and accessories best suited for economical recovery of samples.

(16) GEAR MOTOR: *U. S. Electrical Motors Inc.* has a new bulletin on the type GW Syncrogear motor, showing phantom views of inner construction and operating detail.



(17) SAFETY: A booklet containing an up-to-date list of American Safety Standards has been published by the *American Standards Association.* Comprehensive subject index and brief commentaries on the standards helps in selecting proper documents. Nearly all of the 160 American Safety Standards listed are widely used, and some have been adopted as a basic part of governmental codes.

(18) DIAMOND BITS: A 16-page booklet from *Sprague & Henwood Inc.* tells the story of diamond bits from selection to setting. Various sizes and types of coring and non-coring bits are listed, with descriptions of the various types of alloy matrix available. Eight models of drilling rigs are also illustrated.

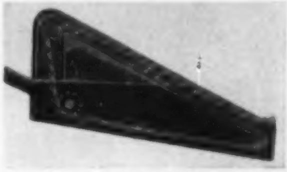
(19) IDEA BOOK: *Black Hawk Mfg. Co.* has released a comprehensive job-picture book showing how hydraulic tools can solve problems for construction, mining, and industrial fields. Sixty-four pages are labelled as containing "1001 valuable short-cuts with hydraulic and hand tools."

(20) MOTOR DRIVES: Allspeed motor drives from 1/3 to 7 1/2 hp are described in a 16-page *Worthington Corp.* bulletin. Drives of upright, horizontal, closed, or skeleton type are graphically portrayed. Dimensions, line drawings, and selection tables complete the presentation.

(21) DIESELS: Bulletin No. 5202 by the *National Supply Co.* gives the specifications and tells the application story of the Superior Model 65 stationary diesel. Six and eight cylinder sizes from 580 to 1500 hp are supplied supercharged, or naturally aspirated, for straight diesel or dual-fuel operation.

(22) CRUSHING: The 36-page Buyers Guide of *Telesmith* equipment illustrates the complete line from crushers to bin gates. The first section features crushers screens and classifiers, the second describes elevators, conveyors, gates and portable crushing plants. Sizes, capacities, weights and power requirements are listed in detail.

(23) CLASSIFIERS: *The Pioneer Engineering Works, Inc.* sand drag type Dehydrator, used in gravel,



quarry and washing plant operations, is described in a new bulletin. Cutaway drawings, specifications and capacities are given.

(24) PUMPS: A reference chart on small pump applications has been developed by *Tuthill Pump Co.* In a one page table this guide lists the various Tuthill pumps, the service for which each is designed, performance, mounting and other selection data.

(25) POTHEADS & TERMINALS: Section 13 of the *Anaconda Wire & Cable Co.* general catalog comprises complete design data, ordering and installation instructions, for all types of potheads and terminals up to 46 KV, in both indoor and outdoor types.

(26) THERMOCOUPLES: Standard thermocouple assemblies and parts are presented in a newly-revised 44-page catalog from *Leeds & Northrup Co.*, which also includes a section on special units for plant and laboratory.

(27) NOZZLES: Anyone responsible for specifying spray nozzles will find a wealth of information in the Industrial Nozzle Bulletin by *Binks Mfg. Co.*

Mining Engineering — September
29 West 39th St.
New York 18, N. Y.

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items indicated.

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
21	22	23	24	25	26	27	28	29	30

Students are requested to write direct to the manufacturer.

Name ________ Title ________

Company ________

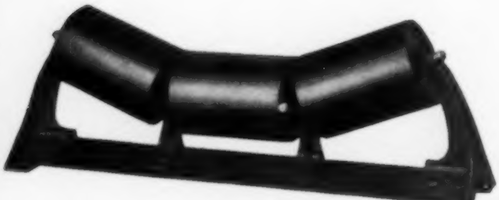
Street ________

City and Zone ________ State ________

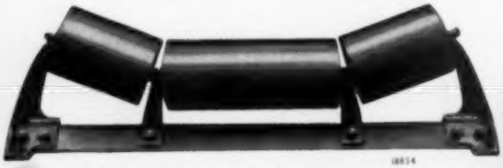
A CHAPTER ON JEFFREY

WHY THEY ARE CHOSEN ON THOSE BIG JOBS . .

No doubt about it . . . there are definite reasons why Jeffrey Belt Conveyors get the call on hundreds of projects where large tonnages must be transported great distances quickly and at low cost.

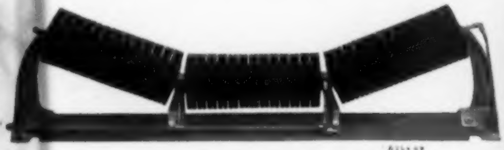


Standard and Heavy Duty 3-pulley roller bearing Idlers for belt widths from 14" to 60".

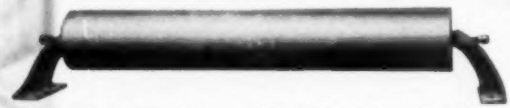


Picking Belt Idlers (with wide center roll) for belt widths from 14" to 60".

Ball Bearing Belt Idlers for moderate service and belt widths from 14" to 36".



Impact Absorption type Idlers with heavy duty bearings, shafts and shells to withstand shocks and heavy loads. Rubber rolls independent of each other—are renewable.



Ball or Roller Bearing Carrying Idlers for flat belts. Furnished for belt widths from 14" to 60".

Standard Ball or Roller Bearing Return Idlers.

Rubber-covered Spool type Return Idlers. Independent rolls for variable spacing across width of belt—provide cleaning action.

Illustration shows an outstanding Conveyor job, complete with self-propelled Stocker (in foreground). Belts covered to provide protection from the elements. Note small photo (upper-right) showing end of Stocker delivering material to stock pile.

Back of it all is sound engineering and experience. Knowing the right type of Idlers to specify is very important. You see, our engineers are specialists . . . know that a Belt Conveyor functions no better than the Idlers upon which the belt carries material.

These same engineers are not afraid to suggest innovations in design to better serve your needs. They have made many big scale, and more often ingenious, applications of Belt Conveyors.

Thus, you can be assured of alert, experienced engineering . . . the ability to build carefully and to design for contingencies—prompt deliveries. Put them to work for you.

1877 — Our 75th Anniversary — 1952

MEET THE AUTHORS

R. U. Jackson *(Conveyor vs. Track Haulage, P. 866)* is the author of a previous work on conveyor systems, which was presented at the Annual Meeting of the Arizona Section of the AIME in December 1951. He has worked for the Carnegie Steel Co., Koppers Co., Fairmont Mining Machinery Co., and the Hewitt-Robins, Inc., the Robins Conveyor Div., for the past 30 years, as engineer, district manager, and manager of the mining div. Jackson is a graduate of Carnegie Institute of Technology, and was born in Steelton, Pa. His present residence is in Clifton, N. J., where in his spare time he devotes himself to hiking and photography interests.

F. S. White (Co-author, *Solids Fluidization Applied to Lime Burning, P. 903)* is a Washington University, St. Louis, Graduate, who now lives in Stockbridge, Mass. He served as a technical apprentice at Mobile, Ala., for the Aluminum Ore Co., and after four years went to Bauxite, Ark. as technical superintendent of calcination and precipitation. After a year and a half, he became technical and operating superintendent of the sintering plant. White also worked for the Dorr Co., and is now with the New England Lime Co. serving as assistant to the president. For relaxation he collects stamps and plays the piano.

Orville Lyons *(Comparative Effectiveness of Coal Cleaning Equipment, P. 895)* attended the University of N. Dakota and the University of Alabama, earning degrees in Mining Engineering and Geology. He later earned a Master of Science in Mining Engineering. A member of Sigma Xi and Lambda Chi Alpha fraternities, he is an accomplished sketcher, in addition to maintaining interests in woodworking and stamp collecting. He has presented six other papers before the AIME. Among the firms he has worked for are: Philadelphia and Reading Coal & Iron Co., as junior engineer; Battelle Memorial Institute, as research engineer; Heyl & Patterson, Inc., coal preparation engineer; and Republic Steel Corp., where he is the manager of preparation. His main interest is preparation and its problems. He has been chairman of the coal div. AIME.

C. C. Wright *(Drainage and Behavior Water Retention Properties of Fine Coal, P. 886)* was born in Southport, England, but received his education in Canada and the United States. He graduated from the University of Washington, Seattle, earning a Bachelor of Science, and later a Ph.D. in chemical engineering. He is a fellow of the National Research Council and has presented several other papers to the AIME. His first employment came with the New England Fishing Co., as a chemist. He later became a research assistant at the University of Washington, and then entered the Fuel Technology Dept., Pennsylvania State College. There he became a professor in 1941 and division chief in 1945. Wright has been chairman of several Coal Div. committees.

E. L. Kinsella *(Co-author of Solids Fluidization Applied to Lime Burning, P. 903)* is presently with the New England Lime Co., living in Pittsfield, Mass. This is his first paper for the AIME. He holds a Bachelor of Chemical Engineering from Rensselaer Polytechnical Institute. Kinsella served with the Army Air Corps for five years, following graduation. Upon his return to civilian life in 1946, he was employed by New England Lime. He is a ham radio enthusiast, and spends all the time he can spare at his hobby.

Canadian owned Lake ore carriers will be used again this year in an attempt to make up the deficit caused by the two months steel strike. Buffalo mills need between 5 million and 6 million tons of ore before the Lakes navigation season ends.

Mutual Security Program aid will help reactivate Austria's largest iron ore mine. The Erzberg development is one of the most important in Europe. MSA announced approval of $500,000 in supplemental financing of equipment. Following the war, the Russians stripped the Erzberg and Radmar deposits of equipment.

The Carborundum Corp. is going to build a $2.44 million plant at Akron, N. Y., near Niagara Falls for production of rare metals used by the Atomic Energy Commission. A subsidiary, the Carborundum Metals Co., Inc., will produce zirconium, hafnium sponge metals.

The pyrochlore rich Sukulu Hills in Eastern Uganda will be mined under a pool arrangement including Tinto Mines, Monsanto Chemicals and Frobisher Mines. The combine will be known as the Uganda Development Corp. Pyrochlore is used in the manufacture of heat-resisting alloys for jet engines.

The way is being paved for the recovery of manganese from low grade ores with a contract between Defense Materials Procurement Agency and the Manganese Chemicals Corp. The pact calls for construction of a plant for treatment of at least 200 tons per day near Riverton, Minn.

Atomic energy is being used by Union Carbide scientists at Oak Ridge National Laboratory for the detection of impurities in ores, metals and other materials. The impurities are made radioactive by placing the material in the Oak Ridge graphite reactor. The exact quantity of impurities can then be measured.

The United States and Portugal will share the cost of mineral exploration in Portugal's two largest colonies, Angola and Mozambique. Mutual Security Agency has earmarked a maximum of $1.3 million, with Portugal paying the part of the expense that can be paid for in Portuguese currency. The colonies to be surveyed are close to cobalt, lead, zinc, tin and manganese producing areas in Katanga and Northern Rhodesia.

Canada's mining industry contributed $1,045,000,000 to the nation's total production, in 1950, according to figures just released by the Canadian Government. It represents a 15 pct jump over the 1949 output. While part of the increase was due to higher prices, most of it can be attributed to increased mineral shipments and production.

Howe Sound Co.'s new cobalt refinery at Garfield, near Salt Lake City, is being run through operational tests, with no date set for full scale production. The plant is equipped to operate under Chemical Construction Co.'s process for the recovery of base metals using autoclaves and a radically new chemical method of extraction.

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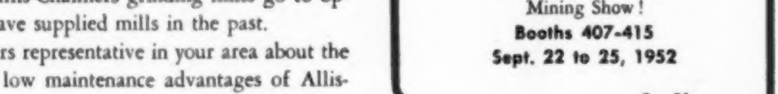

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Michigan Tech Lab Plans Moving Ahead

Establishment of a Bureau of Mineral Research at the Michigan College of Mining and Technology, initiated by the State of Michigan with legislation in 1951, is moving toward realization.

The Michigan Legislature appropriated $24,000 to cover preliminary costs of plans and specifications for "engaging in a broad, widespread study of the beneficiation of low-grade ores, of ore dressing, and all the problems of metallurgy that enter into such a program."

Ralph C. Calder, of Detroit, who designed Michigan Tech's campus buildings, is preparing plans for the proposed Ores Research Building. Members of the college staff, Professors F. J. Tolonen, W. E. Keck, and Research Engineer Paul Jasberg are assisting in preliminary planning.

C. Harry Benedict, retired chief metallurgist of Calumet and Hecla and a member of Michigan Tech's Board of Control, is assisting with technical details. Professor Nicholas Manderfield, head of the department of Mineral Dressing, was assigned to visit other research installations and suggest plans for the Bureau.

M. V. Bomi Hills, built for Liberia Navigation Co. (States Marine Corp.), will go into service carrying ore from the Bomi Hills mine in Liberia to Republic Steel Corp. Able to carry more than 20,000 tons, the ship can carry oil on return trips to Africa. The picture was made during trial runs on the Firth of Clyde near Glasgow.

Liberia Seeks New Contract for Bomi Hills Mine

Liberia wants a bigger share of the profits coming from the Bomi Hills iron ore development. The project is operated by the American-owned Liberia Mining Co.

President William V. S. Tubman told Liberia Mining that 5¢ per ton shipped, Liberia's share, is not enough. He wants profits shared on a 50-50 basis. Operations at the mine, located about 42 miles northwest of Monrovia, Liberia's capital, began about a year ago. Libminco is reported to have been shipping at a million-ton-per-year rate. Most of the tonnage goes to Republic Steel Corp., major Libminco stockholder.

Tubman told U. S. stockholders that 1945 contract is "unjustly weighted" in favor of the company. There is reportedly no ill feeling in the matter. Liberia Mining stockholders are said to have felt contract renegotiation was inevitable, with Liberia deserving of a better deal.

Amazon Region Hit By Gold Fever Epidemic

Brazilian government officials have been showing a growing concern with what adds up to a small amount of gold and a great amount of enthusiasm. Gold fever has reached such a pitch that men are leaving normal occupations throughout the Amazon region to seek gold in the far northern territory of Amapa.

Gold was discovered last May along the Sao Domingo, a stream named by the prospectors who made the original strike. The real extent of the gold deposits cannot be ascertained, but authorities apparently consider it a minor discovery. It is not the first strike in the general area. It has been reported that in the first two months since the discovery about $150,000 worth of gold has been realized. Despite advice to the contrary, thousands of workers are rushing to the scene, paying as high as $250 for transportation up the Jari River. A shortage of boats has helped keep the number down somewhat.

Food Prices Rise

Prices for food and other necessities have risen drastically in the region. Because of the remoteness of the Sao Domingo, authorities know nothing about the real situation. The government has been trying to build a stable economy in the Amazon region, based on forest products. The program is certain to suffer, with a one million-tree rubber plantation at Magazao about to be abandoned because of lack of workers.

Arrow indicates the center of the area where the Brazilian gold rush has been concentrated. The region is extremely rough, and transportation is largely by boat at exorbitant prices.

New Uranium Sources Give U. S. Atom Lead

The United States Atomic Energy Commission is tapping new uranium sources that promise to give the nation a long lead on Soviet Russia in the atomic arms race.

An agreement has been reached with Australia to supply ore to the U. S., by-product uranium from South African gold mines is expected to come to the U. S. soon, and Canada, long a supplier of the metal to the U. S., is putting in additional milling facilities at the Eldorado mine at Great Bear Lake.

Claiming has been completed in the world's first uranium rush in the Canadian north country near Uranium city. Hundreds of prospectors and geologists, on foot, by plane, and by canoe filed claims in what was a far cry from the legendary Yukon and California gold rushes of the past. Conduct of the stakers would have disappointed a Hollywood movie director, according to witnesses.

Uranium ore was first discovered in Saskatchewan's Lake Athabaska area in 1930. Both Canada and the provincial Governments kept the discovery secret until 1949.

(News continued on page 845)

UNDERGROUND, OR . . .

IT'S JOY EQUIPMENT

Right: The Joy Drillmobile, a twin-boom, self-propelled, highly maneuverable machine, gives you maximum footage at least cost per foot of hole. Features Joy Hydro Drill Jibs for fast, accurate hole-positioning, and remote control.

Above: For high-production loading and haulage of rock and ore, Joy teams of trackless loaders and electric or diesel shuttle cars get the call underground.

Right: Complete range of Joy Stopers includes the new S-91T, with telescopic feed. Requires fewer steel changes, gives more time for drilling.

Below: Joy Wagon Drill specially adapted to drill at any height from toe-holes to horizontals 9' high.

Above: Joy Hydro Drill Jibs are versatile units; can be mounted as required to suit individual needs.

Left: The Joy HS-15 high speed drill for underground blast holes, or core drilling to 500'. Compact and easy handling, with "in-line" vibrationless drive.

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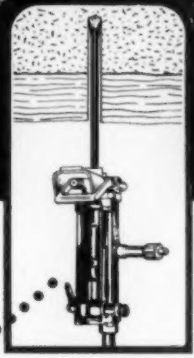
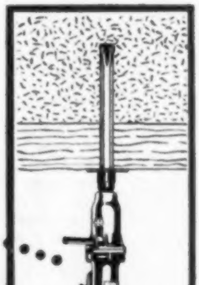


RP-38 "SHORT-LEG" STOPER, for drilling bolt holes and driving wedge type roof bolts. The light weight and excellent balance of this ultra-compact stoper permit easy handling in seams as low as three feet. It has positive, automatic rotation for fast bolt-hole drilling and plenty of power for driving the bolts into the hole. For extra-heavy drilling, the slightly larger Model RP-48 Stoper can be used.



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Ark. Manganese Project Seen Boosting U.S. Supply

A contract between Westmoreland Manganese Corp., and Defense Materials Procurement Agency for mining manganese deposits near Batesville, Ark., is expected to increase annual domestic production of the metal by 52,800 tons.

DMPA loaned Westmoreland $3.8 million to finance the project, with the company putting up $775,000. The money will go into construction of a plant, purchase of land, and operating capital. Jesse Larson, DMPA administrator, said, "this advance, for which 4 pct annual interest is charged on the unpaid balance, will be liquidated as the ore and concentrate is delivered to the governemnt."

For the first four months the Government will pay $167 or $175 per ton of processed ore, based on the type, up to 8000 tons. For the following six months the price will be $137 or $142 per ton. After that, price will be adjusted to operating conditions.

The ore will be extracted from manganese bearing clay underlying the area. The sink-float process will be used to recover the ore after it and other materials have been washed from the clay. The plant is expected to have a capacity of 6000 gross tons per day, producing 240 tons of high grade manganese concentrate for daily shipment to government stockpiles. Mining will be by open pit.

Yugoslavian Tungsten Mine to Start Work

Yugoslavia's first tungsten mine and separation plant is almost completed, with production of concentrates scheduled to begin toward the end of 1952.

The mine is also expected to produce some gold. The tungsten concentrates will be processed at Sibenik to produce tungsten. The mine is located near Neresnica in eastern Serbia, where ore discoveries were made in 1949.

Mine equipment and the separation plant were obtained with aid of an American Export-Import Bank loan.

Bradley Mining Co. Closes Stibnite Mine

The only major antimony mine in the U. S., at Stibnite, Idaho, is being closed by the Bradley Mining Co., because of slackening world demand.

Attempts to interest the Defense Materials Procurement Agency have failed because of necessity for Munitions Board authority to purchase the material. The Munitions board has indicated "no interest." About 20 pct of the nation's peace time antimony supply was accounted for by Stibnite.

Calumet & Hecla Agrees To Reopen Osceola Mine

The big Osceola mine of the Calumet & Hecla Consolidated Copper Co., is scheduled to be reopened under an agreement between the company and the Defense Materials Procurement Agency. The mines are in Houghton County, Mich.

The mine was closed in 1931 and is completely flooded. The Conglomerate, an interconnected mine shut down in 1930, is also flooded. The water is highly corrosive, necessitating the use of special pumps and pump columns. Freeing the Osceola of water will require pumping down to 3600 level, about 2200 ft below the surface. Calumet and Hecla engineers say it will take 530 days to do the job using twenty-three pumps.

Once the mine has been cleared, production is slated to be 7125 short tons of copper per year until June 30, 1962, or until a total of 53,000 short tons have been mined. DMPA has guaranteed the company a negotiated floor-price of 25.25¢ per lb, five-eighths of a cent over the ceiling.

Calumet & Hecla is financing the reopening of the mine with its own funds. Two shafts will be rehabilitated. Total cost is expected to reach $6 million. Submersible pumps having a capacity of 13 million gal per day will be used. Most of the mine equipment will have to be replaced because the original machinery was scrapped during World War II.

NPA Seeks Greater Titanium Production

The National Production Authority is tackling the job of increasing the supply and use of titanium, with an expected annual production of 8000 tons by 1955.

Principle problem involved in using the metal right now is the prevailing price, about $5 a lb. Production for 1952 is estimated at about 2000 tons, and for 1953 at 4000 tons. Potential military requirements are expected to be well over the anticipated supply, but at present industry representatives state that current orders are lacking in strength to keep plants in operation.

In addition to increasing the quality of titanium products, it has been agreed that technological development will have to be condensed from 25 years into five or six years.

436 Mineral, Metal Expansion Projects

The Defense Minerals Procurement Agency approved 436 expansion projects during the fiscal year ending June 30 to increase the supply of metals and minerals.

The agency said that industry made an outlay of nearly $672 million, but no breakdown on the amount spent by Government was available. Jess Larson, agency head, emphasized that the program was a joint venture and "one of the finest demonstrations of Government-industry teamwork that the country has ever seen."

The greatest number of projects approved for one metal was 96 for the expansion of iron ore production by 30.73 million tons a year. The list also included 53 projects for copper, 26 for ferro alloys, 92 for lead, zinc and cadmium, five for tin, three for titanium and 131 for more than a dozen non-metallic materials.

Also included were 23 projects for the improvement of existing aluminum plants. The list did not include projects of the Defense Production Administration.

Huge Canadian Power Plant Starts Operation

One of the largest hydroelectric power projects in North America has gone into operation at Chute Du Diable on the Peribonca River in Quebec. The plant was built by the Aluminum Co. of Canada at a cost of $30 million.

Each one of the plant's five generating units is capable of producing almost 70,000 horsepower. The units are being opened at three-week intervals. A twin plant, Peribonca No. 2, is expected to be in operation early next year. The added power is expected to greatly raise aluminum ingot production in the Saguenay region, some sixty miles from the power plant.

DMPA Sets Goals For Steel Expansion

Defense Materials Procurement Agency wants iron ore output to reach 147 million tons by 1955 to support the steel industry expansion now underway. In addition to iron ore, standards were raised for ore transport and alloying metals.

The increased goals are aimed at supporting a steel industry capable of producing 122 million tons steel ingot per year by 1955. The iron ore goal does not include taconite. Eight million tons of the iron ore expansion goal of 57 million tons is earmarked for replacement of production from certain depleting mines. The increase is to be provided by domestic mines or foreign mines operated by American interests.

Six Ore Carriers

In addition to the iron ore increases, a goal of six more Great Lakes ore carriers was set. The 20,000 ton capacity carriers will either be constructed or converted from existing shipping, with the program scheduled for completion by mid-1954.

Manganese ore supplies are expected to increase by 630,000 tons per year over the 1950 production of 1.87 million tons from domestic mines and imports. The total desired supply in 1954 is 2.5 million long tons. Part of this expansion is also in progress.

Goal for columbite and tantalite is 3 million lb by 1954. DMPA also wants a galvanized sheet and strip steel production of 1.3 million net tons by 1954, doubling the 600,000 net ton pre-Korean capacity.

Sulphur Exports Start From Ecuadorian Property

Sulphur exports from the Tixin Mine of the Ecuadorian Mining Co. have begun, in addition to supplying local needs. Chemical Plants Corp. of New York has the contract for exploitation of the deposits for 20 years.

The lands involved in the pact are 100 miles from Guayaquil at an elevation of 8500 ft in the Andes Mountains. The land was acquired by the Ecuadorian Government in 1902 from the Roman Catholic Church. They were then transferred to the Public Welfare Board of the state of Chimborazo.

Experts have adjudged the deposits among the most favorably situated in South America. An additional 1000 sq miles were recently granted to Ecuadorian Mining for exploration and exploitation.

Valuable deposits of sulphur have also been reported in the states of Carchi and Cotopaxi and in the Galapagos Islands.

Mines Bureau Develops Thallium Recovery Method

The U. S. Bureau of Mines Intermountain Experiment Station at Salt Lake City, has developed two methods for the recovery of thallium, a metal of increasing importance.

The methods have resulted in high recoveries from white arsenic and from lead smelter flue dust. During World War II the metal was used in lenses of optical equipment employed for signalling by infrared light.

Using smelter products supplied by the American Smelting & Refining Co., Bureau metallurgists retained in residue more than 99 pct of the thallium from a white arsenic containing 96 pct arsenic trioxide and 0.21 pct thallium. Ninety pct of the thallium was recovered from lead smelter flue dust, in thallous chloride crystals, purified to a final product of 99 pct purity. Overall recovery of thallium in refined crystals was more than 85 pct.

Volatization was used for the white arsenic and cyclic leaching for the lead smelter flue dust.

Japanese, U. S. Firm In Nickel Mud Venture

Joint plans for the construction of a 250 ton per day pilot plant in Japan for the production of nickel mud are being worked out by Newmont Mining Corp., and the Nihon Yakin Kogyo K. K. (Japan Metallurgical Works).

Newmont has agreed to loan \$2 million to Nihon Yakin Kogyo for building the plant on condition that all nickel mud produced go to the new smelter the American company has built at Vancouver. Newmont is now preparing a design for the pilot plant. Using an undisclosed, special technique, the nickel mud is expected to be produced at a very low cost, according to officials of the Japanese firm. Le Nickel, a French firm, will supply ore from New Caledonia.

Mobile to Handle Venezuela Ore Shipments

Completion this month of the bulk handling plant on the Mobile, Ala., docks will increase the port's ability to handle incoming ore shipments. The expansion will probably mean that much of the ore mined by U. S. Steel at its Orinoco Mining Co. development in Venezuela, will come through the port.

Ore shipments from Venezuela are expected to begin when U. S. Steel finishes installation of facilities at the mine. A large part of the ore is earmarked for Birmingham steel mills. Equipment being moved up the Orinoco River to Puerto Ordaz includes house trailers, pipe, tie plates, dredging machinery, railroad cars, and other items.

Tin Negotiations Remain At Standstill In Washington

Negotiations between the Reconstruction Finance Corp. and the Banco de Minero of Bolivia for the purchase of Bolivian tin have made little or no progress in more than a year of talk.

With the arrival of a new Bolivian ambassador, Victor Andrade, to the U. S., it is expected that the negotiations will continue to be bogged down. The new envoy, however, has expressed optimism.

The Bolivians have been holding out for a price of \$1.30 per lb, while the U. S. has offered \$1.215, the same price paid to other tin producers. The RFC is a buyer and seller of this metal for the U. S., although private importation is now permitted, while the Banco de Minero is the representative of all Bolivian producers under the new regime in that country. At this point, elementary matters such as duration of the contract, have not been reached. While Bolivian tin producing facilities are still in private hands, nationalization of the industry is one of the proclaimed objectives of the new government. The committee investigating methods for nationalization has more than a month to report to the government.

Should nationalization take place before an agreement is reached, the U. S. State Department may have to enter into the proceedings. Recent spot sales to the RFC have been made at the \$1.215 price.

South Korea Plans Double Tungsten Output

A $2 million project has been started by the South Korean Government to double production of tungsten ore. Monthly production now averages about 250 tons.

Tungsten ore is South Korea's chief dollar earning commodity, with all production going to the United States under an exclusive contract agreement. When the project is completed sometime next May, South Korea will be able to supply one-third of the world's needs for tungsten, according to the Ministry of Commerce and Industry.

Modify EIMCO Muckers To Fit Loading Job

Members of the AIME attending the Summer Meeting of the San Juan Sub-section held at Telluride, heard how two Eimco Model 21 mucking machines were redesigned to do the loading job involved in an inclined shaft mucking operation at Ophir Mine, in the Ophir district, Utah.

Don Willie, speaking before the group, described how U. S. Smelting sank a 23° to 26° incline. The winze followed hard mineralized limestone beds down dip a distance of about 1500 ft. Specially designed equipment increased monthly footage in the incline from about 65 ft to 120 ft.

Two Eimco model 21 mucking machines operating on parallel tracks in 8 ft x 12 ft incline worked at vertical angles considerably greater than 20°. Two hundred lb counterweights were mounted on the machines to minimize derailing. Drums were mounted inside the rear wheels instead of outside to reduce tipping forward. Cables were anchored on the ties between the rails and wound on the mucking machine drums inside the rails and passed under the skip behind the loader.

Caution Required

If the operator used caution while loading coarse muck the weight of the skip served to keep the mucking machine from tipping. The cables were anchored every 40 ft down the incline. One hundred psi air pressure was needed for loading at such steep angles. Mucking time in the cycle of operations was about one shift.

The equipment, originally designed for operation at vertical angles up to 26° behaved badly at vertical angles of 27° or 28°.

JOHN L. LEWIS is leading with his left again, in preparation for another right. Where and when the punch will land is pretty well known, but its affect upon the coal and steel industry is uncertain. It may be that Lewis has chosen the wrong time to begin wage talks. The above-ground coal stockpile is pretty healthy, and the general economic condition of the industry could make a Lewis onslaught a little less deadly than usual. Coal is in one of its less stable financial moments, and operators might be inclined to fight Lewis rather than give in to demands they feel they cannot meet and survive. Lewis has given notice of contract termination as of September 20. It is a possibility that Lewis wants something other than wage increases. He's negotiating with northern mines exclusively, however, following the pattern of last year's coal talks. Once the northern coal operators were in line, Lewis made terms with southern and western mine owners without trouble. Harry Moses once again sits across the table from Lewis. So far only the two of them know what the demands are, and neither is talking.

The 10 day work stoppage memorial called by Lewis is the best indication of the kind of tactics he intends to use. The miners stayed away from work starting August 23, with an estimated 10 pct of the 84 day coal supply eaten up in the process. It's a tactic that the mine union boss has used before with telling effect. As usual, Lewis is taking the offensive. The 334 miners cited as killed by the United Mine Workers Journal is all the justification Lewis needs for his memorial call.

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NEWSPAPER headlines announcing the end of the world sulphur shortage found rebuttal in at least one segment of the U. S. sulphur industry. Langbourne H. Williams, Jr., president of Freeport Sulphur Co., was quoted in newspapers throughout the country as saying "although sulphur consumption is still under government limitation, all but a few customers in this country are getting all the sulphur they need, and the situation abroad has also improved. The outlook for the future is extremely encouraging. There is enough new production in sight to dispel the threat of continuing shortage . . ." Williams said that estimates are for an additional 4 million long tons of sulphur per year by the end of 1955, to be derived from almost 100 projects in the U. S. and other free world countries. Estimated 1951 free world production was 12 million tons of sulphur in all forms.

Viewing the statement critically, Paul Nachtman, president of the Mexican Gulf Sulphur Co., said, in a direct reply to Williams' pronouncement, "It would be heartening to believe that with the great new discoveries of sulphur on the Isthmus of Tehuantepec, Mexico, and elsewhere . . . the ever increasing demands for it can be met. . . . Even though the new Mexican sources are extensive, . . . a study of economic and industrial experience shows that with each progressive step in the development of new materials and products . . . and adoption of new methods in their production, *the demands for sulphur mount steadily higher and for the next several years at least will be beyond the visible or potential supply.*"

Nachtman quotes the Paley report:

'That the great Texas and Louisiana deposits of pure brimstone are dwindling and that the costs of new sulphur discovery are rising are facts known and appreciated by industry, which has already been severely shaken by the thought of failing, cheap, sulphur supplies . . . the need for sulphuric acid will probably not abate during the next quarter century and it is entirely possible that if we do not move toward recovering some more sulphur than we do now, *a new and heavier sulphur stringency may be upon us again.*'

It becomes obvious from a quick look at these divergent opinions that the debate stems from the difference in time measurement. Williams considers the next three years as the critical ones in world economic history, while Nachtman emphasizes the needs of the next quarter century. One can only guess at the rate of acceleration of world consumption within the near or distant future. Standards of living rise and fall, based on the caprices of market, money, ethnic predilection, and politics. Thus, no one can presume to guess with true accuracy what the sulphur consumption of the world will be within a given time limit.

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THE Senate Preparedness Subcommittee and the Malayan Tin Bureau started throwing things at each other recently. The debate grew out of the Johnson report, which charged that Malayan tin producers were responsible for the rise in tin price between June 1950 and February 1951. The Malayan tin producers lost no time in denying that a cartel existed, and that anything but U. S. stockpiling of tin was responsible for the rise in price. It was also pointed out that the agreement between Britain and the U. S. for the purchase of tin did not benefit anyone but the U. S. Reports from Britain indicate that British commentators feel that once again the U. S. has managed to get the most for the least. As for the Malayan tin producers, they deny that Malayan tin is easily produced and that all of the ore is high grade. They also claim high cost producers have been forced to suspend operations because of the present price of tin. The Government has been the sole purchaser of tin in the U. S. In recent weeks a number of sources called upon the Government to return tin purchases to normal commercial channels, allowing users to make their own deals directly with producers.

The government has already begun to step out of the tin buying business. The Reconstruction Finance Corp. will continue to buy on a limited basis, however, under existing contracts, in addition to concentrates for its Texas City, Texas, smelter.

Constant danger of attack from communist guerrillas has curtailed exploration efforts in Malaya, according to the Tin Bureau. Reports of communist

assaults against working mines have been reaching the U. S. with regularity. Despite the presence of Commonwealth troops in Malaya, many tin mine employees have been killed and equipment and installations damaged. One American tin company, Pacific Tin Consolidated Corp., ranks fifth among Malayan producers. Its president, Norman Cleaveland was among those who denounced the subcommittee report. He expressed the opinion that the report can have a bad affect on Asian relations, now in one of the most precarious states in history.

The Tin Bureau has denied that there is any shortage of tin. It has also denied that any conscious cutback in production has been made. It wants tin placed on the free market in the United States, with the accompanying opportunity for price to seek its natural level in accordance with the machinations of free trade.

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GERMANIUM, once the metals industries' unwanted step-child, has suddenly blossomed into importance. As late as 1951 textbooks failed to mention it and the metal was considered no more than a nuisance that interfered with the electrolysis of zinc. Eagle-Pitcher is the only producer of Germanium, used in power rectifiers and the new transistors. Its basic quality is that its ability to carry current can be changed, and a current flowing through it can be controlled. Germanium is sold by Eagle-Pitcher for about $340 a lb, not bad for a metal that only a few years ago was totally ignored. Eagle-Pitcher gets germanium, cadmium and gallium by roasting the zinc concentrate to convert the sulphides to oxides, then adding coal and salt to the crude zinc oxide and sintering at a high temperature. The oxides of germanium, cadmium and gallium are converted to chlorides. Cadmium and germanium chlorides volatilize, and are gathered for additional processing. The important step is the distillation of germanium tetrachloride.

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PRESIDENT TRUMAN signed the Neely Federal Coal Mines Inspection Act, but not without finding fault with the law. He pointed out what he considers to be five failings in the bill. In signing it however, the President called the bill a "significant step in the direction of preventing the appalling toll of death and injury to miners in underground mines."

Principal criticism offered by Truman was that the law had no control over coal mines employing fewer than 15 men. These small mines are not required to comply with mine safety provisions, even if disaster is imminent. Truman also pointed out that the law is directed against major disasters from five causes—explosion, fire, inundation, mantrip or man-hoist accidents. But these causes have been responsible for only 7 pct of accidents in the last 20 years.

The President claimed that under the new law replacement of dangerous electrical equipment and faulty ventilation systems, cause of most recent disasters, may be evaded through several exemptions. It was also claimed enforcement of the law is made difficult by complex procedural provisions. Finally, he found fault with enforcement vested with the director of the Federal Bureau of Mines, rather than with the Secretary of the Interior.

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A NEW word is coming into the vocabulary of German labor. It's *co-determination*, and its impact may be felt throughout the world someday. It might get out of hand and grow into a monster. *Co-determination* means joint management of industry by capital and labor. A law passed by the West German parliament a year ago introduced the system into every coal, iron, and steel plant in the Ruhr employing more than 1000 workers. Recently, parliament extended the law to almost every private enterprise employing five or more persons. Labor has a 50 pct voice in coal and steel, but thus far, it has been a marriage in name only. Labor and management seem to be probing for weaknesses and strong points. The labor representatives have been sitting with regular board members with the number of labor directors equal in power and number to management. Neither has made a real attempt to challenge the power of the other. The law provides that deadlocks between labor and management directors shall be decided by a neutral. The power of the Ruhr barons, taken away by the Allies, has been redeposited with the labor unions. They have the means for determining profit and loss, and general financial conditions. Whether they will use the power wisely is to be seen.

Several things could happen. Conceivably the old cartel, destroyed by the war, could be replaced with another, composed of labor and management. Possibly labor and management will meet at loggerheads on every important issue, jamming up production and economic development. The West German parliament, faced with an increasing demand for nationalization of the Ruhr, may have arrived at the only answer. Europe's economy is changing quickly, and the results are predictable only with a great many reservations.

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ANTHRACITE is turning glamorous, with 24 tons of it going into the design of the bottom of the new fountain pool in front of the Secretariat Building of United Nations headquarters in New York City. Originally, black stones from the Island of Rhodes were to be used in alternate bands with Vermont Marble for the black and white serpentine motif. The stones did not arrive for the pool opening, and anthracite was substituted. Lehigh Navigation Coal Co., Lansford, Pa., supplied the coal which was mined in the Panther Valley Fields. Two truckloads were required for the job. The coal supplied the black and white contrast as well as the originally planned stones would have.

MINING ENGINEERING

EDITORIAL

DON'T LET IT DIE

THERE have been two important accomplishments of the Truman administration; the Hoover Commission report on inefficiency and waste in government and the report of the Paley Commission on the natural resources of the United States. Both reports are in the interest of the American people and should transcend the boundaries of political parties. Both reports are starting points for the formulation of policies which must be continuously revised to meet changing conditions. Regardless of the outcome of the election the principles embodied in these two great documents must be kept alive.

The Paley report is right in our own bailiwick of mineral raw materials and it behooves us to study it carefully as our responsibility to posterity. To those in the mining industry who have limited their field of observation to moving rock or to metal prices it may come as a surprise that our production of such staples as copper, lead, and zinc has fallen behind increased requirements to the point that substantial tonnages must be imported. Iron ore is next on the list and everybody knows that such metals as tin, tungsten, chromium, manganese, nickel, and cobalt are imported almost exclusively. The crisis in raw materials pointed up by the Paley report has been reported annually by Elmer Pehrson and others to small gatherings at the Mineral Economics Division sessions at the Annual Meeting for as long as we can remember. In September 1949, the United Nations Economic and Social Council held an international resource conference at Lake Success which disclosed these same grave conclusions:

► Consumption of raw materials is multiplying at an alarming rate, faster than discoveries or replacements are being made.

► Production has been wasteful and utilization lavishly beyond engineering and practical requirements.

► Industrialized nations are depleted and many raw materials consumed by industry are not found in the industrialized countries.

► Nations formerly exporters of raw materials are increasingly building their own consuming industries.

► Political forces in the world are limiting access to raw materials.

The Paley Commission selecting the quarter century, 1950 to 1975, to project raw material requirements for the United States, pointed out obstacles to meeting these requirements, and made recommendations for procurement. It is not important that we disagree that cobalt consumption will reach 344 pct of present, or fluorspar 187 pct when iron and ferroalloys are only expected to increase 75 pct. What is important it that the fabulous appetite of the United States for mineral raw materials has consumed more since World War I than was consumed by the entire world in all recorded history prior to 1914. And, unless the people of the United States are satisfied to remain at their present standard of living (which is unthinkable) this consumption is correctly plotted by the Paley Commission as continuing to expand.

If the U. S. is to continue on a course of progress, it must procure raw materials wherever it can at the lowest possible cost. Natural resources are clearly an area in which the broadest cooperation between industry and government must exist to achieve efficient exploitation, economic consumption, and expansion of sources of supply. The message of the Paley report requires action comparable to that of the North Atlantic Pact. Its implication is as significant as military preparedness. It is a world problem. It is a mining engineering problem. Don't let it die!

The conveyor delivering to sample plant and fine ore storage bins at the Deming mill of American Smelting and Refining Co. The flotation plant is in the background.

Deming Mill — A Materials Handling Problem Solved

by Norman Weiss and H. W. Kaanta

CUSTOM ores reach across the New Mexican mesa to American Smelting & Refining Co.'s new lead-zinc mill at Deming. The influx of ores justified increasing mill capacity from 12,000 tons per month to 20,000 tons per month with the installation of a second milling section. Deming mill has met a complex material handling problem with the successful instrumentation of modern machinery and knowledge. A tight, compact plant does a big job with the aid of the latest ore moving equipment.

Location and Layout

The mill is located three miles northwest of Deming in the tumbleweed covered valley of the Mimbres River, 30 miles north of the Mexican border. Deming has a population of 4000 and is about halfway between Lordsburg and Las Cruces and 54 miles southeast of Silver City. The AT&SF and the SP railroads and U.S. Highways 70, 80, and 260, cut across the mesa to the town.

Asarco mill station is east of the AT&SF single track, and north of Deming. The nearly level terrain is covered with fine sand supporting only yucca and tumbleweed. Compacted gravel at depth affords excellent foundation bearing. The climate is semi-arid, but the district is underlain by water-bearing strata. Tailing disposal space is plentiful.

The Deming site appeared favorable because of easy rail and highway access, central location in a mining district, excellent climate, plentiful water supply, space for all present and future requirements, and proximity to a town.

The milling plant is a 250-ft square. The rail spur forms the west side; sampling plant, fine-ore conveyor and bins are on the east; track hopper, primary conveyor, and crushing plant lie on the south; and the mill building completes the north side of the square.

N. WEISS is Milling Engineer, Salt Lake City, and H. W. KAANTA is Mill Superintendent, Deming, N. Mex., both with the American Smelting & Refining Co.

Construction

The mill site was chosen and the land purchased in 1946 and 1947. Design began in 1948, and Stearns-Roger Mfg. Co. of Denver started construction in June 1949 completing work in April 1950.

Construction is steel and concrete throughout, with main concrete floors supported on cellular steel beams. The building is covered by galvanized corrugated iron, attached to the frame by welded Nelson studs, and made dust tight at the joints with roofing paper and asphalt sealing compound.

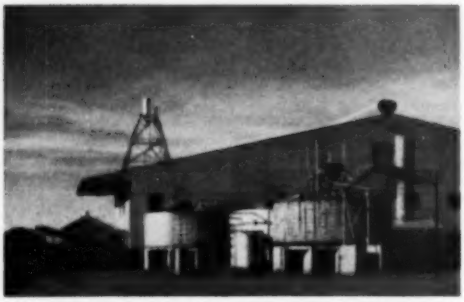
Crushing and Sampling

Custom ores, delivered to the mill by railroad cars or trucks, are weighed on arrival. A Robbins car shaker dumps the ore into a 100-ton capacity steel hopper. Adjacent is a 40-ton truck hopper.

Three 36-in. Pioneer apron feeders supply ore from the receiving hoppers to a 36-in. conveyor which runs to the crushing plant. A tramp-iron detector protects the primary crusher except when handling magnetite containing ores.

A 2½-in. spaced grizzly precedes the primary Traylor 24x36 in. jaw crusher set at 4 in. A Symons 4-ft cone crusher in closed circuit with 48x78 in. Robbins Gyrex screen completes crushing to ⅝ in. The discharges of both crushers drop upon a common conveyor, which, after gaining sufficient elevation, transfers ore to a return conveyor. This return conveyor passes under a suspended magnet and over a magnetic head pulley to the Gyrex screen. Screen undersize is conveyed by a 20-in. belt to the sampling plant. Crushing rate varies from 60 to 90 tons per hour.

Ores containing magnetite and pyrrhotite nullify the magnetic detector and overload the magnets



South side of Deming mill with thickeners outside in the foreground, and concentrate loading equipment on the railroad dock at the left.

protecting the Symons crusher. Attempts were made to cut off the magnets at such times and to post a man on the belts to remove tramp iron; unsatisfactory visual detection of tramp made this ineffective. Another solution was to cut out the suspended magnet and catch the magnetic material at the head pulley, dropping it on the conveyor going to the fine-ore bins. This failed because coarse material, bypassed to the bins, interfered with sampling and feeding.

Finally a method was adopted that protects the secondary crusher without destroying validity of sample or disturbing feeding. Material caught by the magnetic head pulley drops into a small bin. After a batch of magnetite-bearing ore is crushed this small bin is emptied back into the system by a short picking belt. This slow moving belt returns magnetic material to the system while the operator removes tramp iron.

At the sampling plant a Heginbotham sampler cuts the ore stream every 2½ min. A small conveyor positioned by remote electrical control from the crushing plant transfers the sample to one of six 5-ton sample bins. Bins are equipped with a high-level indicator. Samples are reduced in bulk and particle size by two stages of roll crushing and sampling. Fifty to 200 pounds of $-\frac{1}{8}$ in. material reaches the lowest floor of the sampling plant for preparation of assay pulps.

A Robbins motor-propelled tripper sends the main stream of ore from the sampling plant to one of four cylindrical steel bins, total capacity 1200 tons. Sample plant rejects go to these bins by an elevator to the feed conveyor.

The crushing and sampling plant was designed to solve the problems of handling custom ores in large and small lots. Track and truck hoppers were engineered for rapid cleanout between lots. Pan feeders, primary conveyor, and primary crusher are rugged enough to handle larger-than-normal ore.

Integrity of individual ore lots can be maintained only when chutes, boxes, bins, and machinery present little obstruction to ore flow and spill is negligible. A shipper can see for himself that his ore passes through the crushing plant without contamination or loss.

The sampling plant is one of the most complete to be found in a mill of this size and type. The main design objectives were simplicity and accuracy. Good dust control and vacuum cleaning help keep the building and equipment clean.

The Mill

The mill building is 64 ft wide and 139 ft long. Its width can be extended by one-half if the need should arise. Only the south half of the present building was originally equipped with milling machinery. The north half was reserved for the second milling section of 200-250 ton per day capacity, installed in June 1951.

The ground floor contains grinding and pumping equipment. Flotation machines and filters are on the upper operating floor. Reagent feeders and storage tanks occupy a mezzanine. Concentrate thickeners are outside the building.

Ore is drawn from the four 300-ton fine-ore bins by Hardinge constant-volume feeders and chute-and-gate feeders. Three 24-in. collection conveyors on the long centerline of the four bins then carry the ore to ball mill feeder-belts, the central conveyor of the three being reversible. Ore from the first two bins goes to section 1 and from the last bin to section 2, but ore from the third bin may be fed to either section. The four fine-ore bins and the reversible conveyor provide for blending ores to improve metallurgy.

Mill Section 1: This is the original 400-ton section completed in 1950. The collecting conveyors deliver to a 24-in. inclined belt which passes over a Merrick Weightometer and feeds the March 77 ball mill through a combination scoop. Feed rate is about 20 tons per hour. The ball mill works in closed circuit with a 5x25 ft Dorr DSFX classifier. Overflow is at 42-45 pct solids, 8-10 pct +65 mesh, 58-60 pct −200 mesh. A constant-head tank on the mezzanine supplies grinding section water.

Classifier overflow is pumped to flotation by a Hydroseal A-frame pump and standby. Feed is automatically sampled before entering the lead section. Ten No. 24 Denver flotation cells produce lead rougher froth, with cleaning in two No. 21 Denver machines. Final lead concentrate is automatically sampled and then pumped to a 20-ft Dorr thickener.

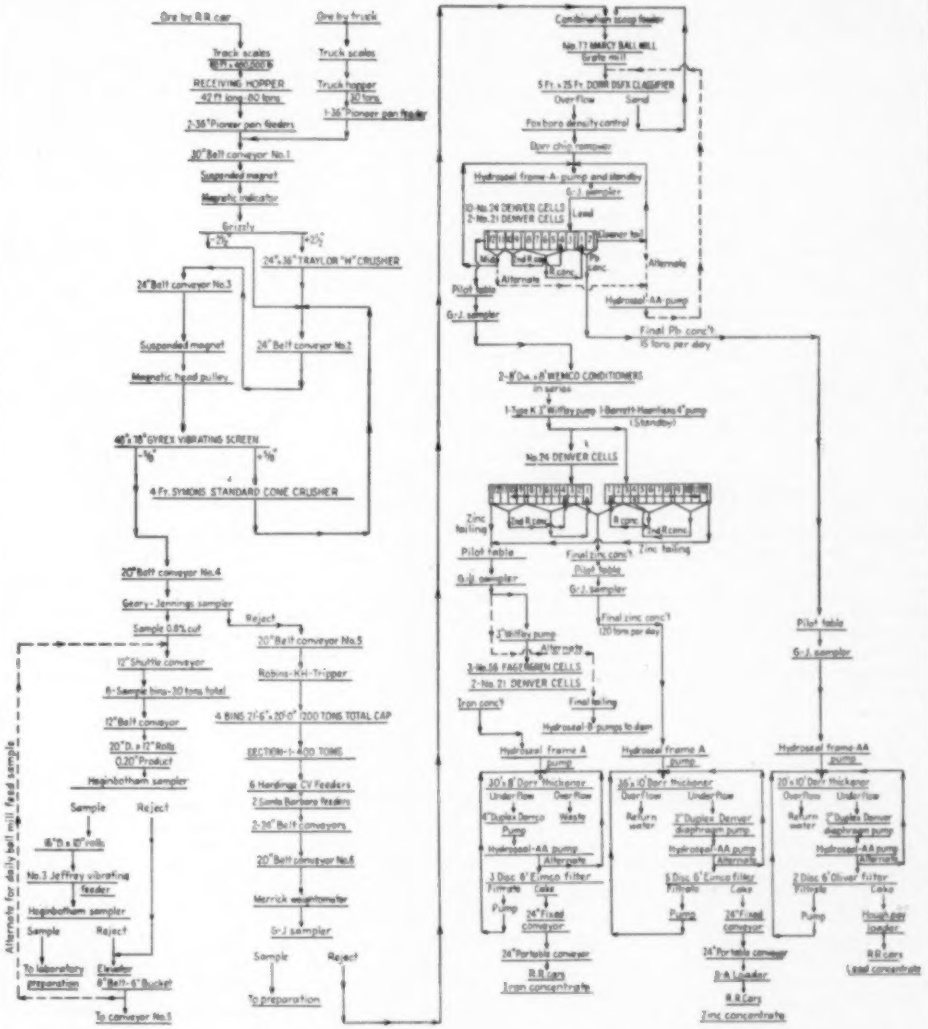
The tailing of the lead rougher string drops to two 8x8 ft Wemco conditioners in series, located on the lower floor. From the second conditioner a 3-in. type K Wilfley pump or a Hazelton "CT" pump elevates the pulp to the zinc section. The zinc flotation section consists of two 12-cell No. 24 Denver machines in parallel. Feed enters the fourth cell of each; the second and third cells are primary cleaners and the first cell in each row is the final cleaner. The froth from these is automatically sampled and pumped to a 36-ft Dorr thickener.

Lead and zinc concentrates are thickened and filtered. The handling of thickened concentrates is a little different from conventional procedure. At Deming concentrates are removed from the thickeners by diaphram pumps and delivered to centrifugal pumps on the lower mill floor. The centrifugal pumps supply filters situated on the mezzanine level with the flotation machines. Two-stage pumping was considered advisable because of the flatness of site.

Less than a carload of lead concentrate is produced per day. Cake drops from filter to storage floor. When a carload is accumulated a portable conveyor, loaded by Hough Payloaders, discharges the cake into gondola cars. Zinc concentrate drops directly from the filter to a fixed conveyor, feeding a portable conveyor, which last in turn feeds a slinger-type box car loader.

Asarco mill siding has two main tracks, with crossover and passing track. The concentrate loading

FLOWSHEET
DEMING MILL



Flowsheet of crushing and sampling plants and of Mill Section No. 1. Equipment details of Mill Section No. 2 are given in text.

track is laid on a 1 pct grade to facilitate car movement to the scale, where cars are weighed and sampled for moisture and assay.

Mill Section 2: The second section of the mill, with capacity of 7000 tons per month, was installed in the spring of 1951. This addition involved relatively little work, since original plant design provided for expansion. Major equipment consisted of new belt feeders, a 7 ft x 36 in. Hardinge mill and 6x20 ft Dorr DSD rake classifier, one 8-cell bank of Denver No. 18S flotation cells for lead roughing or cleaning, an 8-ft Wemco conditioner, one 12-cell bank of Denver No. 24 flotation cells for zinc roughing and cleaning, and various pumps. A 45-ft thickener was added to assist the 36-ft thickener in handling increased production of zinc concentrates. No changes were necessary in the filter section.

Products of both sections are visually sampled on pilot tables, and automatically sampled for assay.

Tailing Disposal: A 600x1000 ft dam site was selected 2440 ft northeast of the mill. The strong prevailing wind is south-southwest and the best site was leeward of the mill on a slightly elevated mesa beyond the Mimbres River. The main problem was to pump from the mill directly to the dam without obstructing the mill yard and the county road or incurring frequent stoppages in the pipe line.

The final tailing from the zinc or pyrite flotation cells goes to one of two cone-shaped sumps. Each sump is connected to a pair of Hydroseal B-frame pumps in series, one sump and pump-pair being a standby. These pumps elevate the pulp to a point just outside the mill and 22 ft off the ground. From here the 5-in. pipe line drops at ⅛ in. per ft until it crosses the county road with 18-ft clearance; then drops at a slightly greater pitch to a solenoid-operated dump valve 740 ft from the mill. From the valve the line climbs 1/16 in. per ft to the dam.

The manually-reset solenoid operated dump valve drains the line of most of its contents when power is cut off the tailing pumps, or when the operator wishes to clean out the line.

One of the two sets of tailings pumps is semiautomatic. The first pump in series is started when the pulp reaches a certain level in the cone, and the second pump in series starts automatically as soon as the first pump cannot handle the job alone. The first pump is speed-controlled by a liquid-level system operating through a hydraulic coupling. The second pump operates at constant speed, but can be altered by a single change in vari-pitch sheaves.

If the pulp threatens to overflow the cone, the operator is warned by horn and light. If at the end of 15 min the level has not dropped, the dump valve opens, draining the line.

The original 4-in. discharge pipe line was replaced by 5-in. pipe when the second mill section was placed in operation.

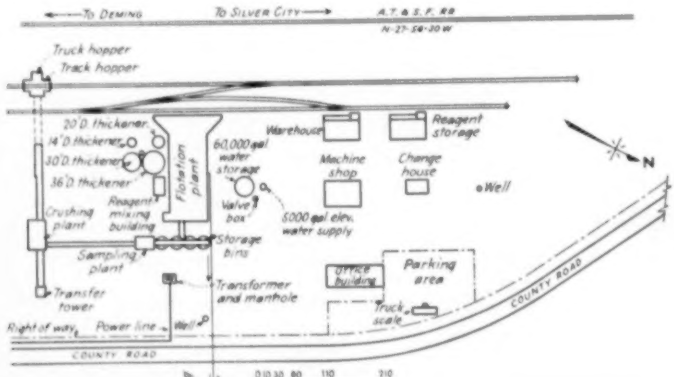
Spill Recovery: Three floor sumps collect mill spills, one for the grinding and lead sections, another for zinc circuit spills, and a third for excess water from the other two. The first two are equipped with Nagle 1½ in. vertical-shaft pumps, which return pulp to the flotation circuits. The flotation floor is a wood grate overlying a self-draining concrete slab laid between steel beams.

Pyrite Recovery

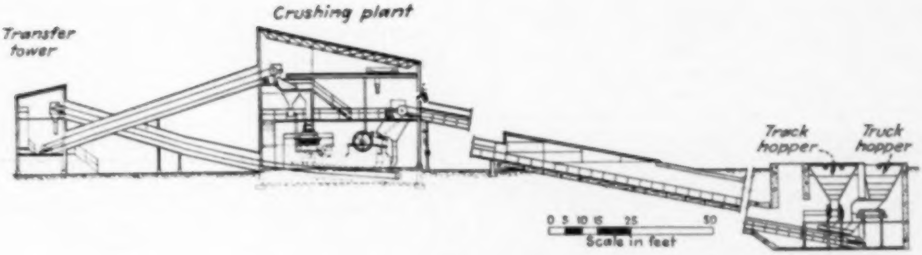
Some of the ores contain a considerable quantity of pyrite. In anticipation of the day that pyrite flotation concentrate might find a market, a section was installed for its recovery. The zinc section tailing, which ordinarily goes to the final tailing pumps, can be diverted wholly or partly to another pump which feeds the pyrite section. This consists of three Fagergren No. 56 cells and two No. 21 Denver cells, arranged in such a manner that (1) the former are roughers and the latter are cleaners, or (2) the froth of all cells except the first may be recirculated to the head of the pyrite circuit, and final concentrate removed from only the first cell. The pyrite section has operated experimentally and is on standby.

Metallurgical Control

Metallurgical control in a mill which buys ores from widely scattered mines is more complex than in a single-ore mill. Since the various lots must be crushed separately for sampling, some means of obtaining a reasonably constant ore mixture for milling is required. The most practical means of blending ores for a plant the size of the Deming mill is



Map of the general layout of the Deming mill showing routing of materials. The main part of the plant, in the upper left is approximately 250 ft square.



Crushing plant section also showing conveyor transfer point and track and truck hoppers. Individual lots of custom ore may be crushed and sampled, while being kept completely separate, without stockpiling or rehandling. Ore goes from the crusher to sampling by a conveyor at right angles to this section.

to use a number of fine-ore bins. Ores of similar character go to one bin, and in that bin the best possible mixture of ores of different origins, but the same general character, is obtained.

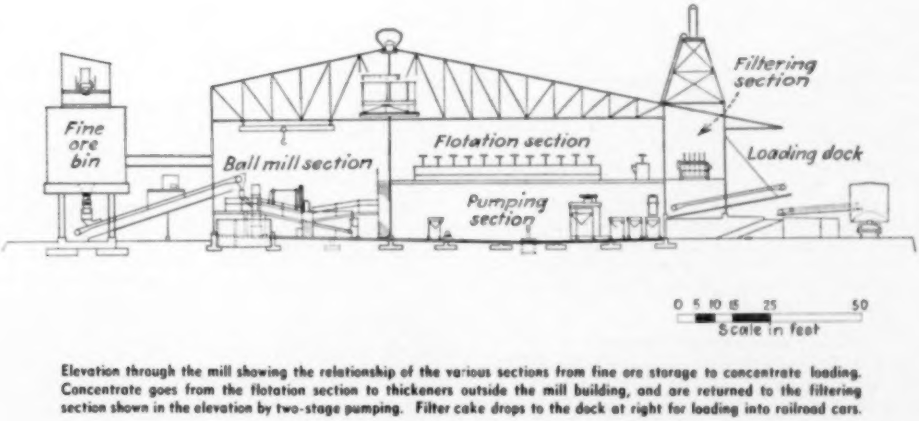
At Deming the four 300-ton fine-ore bins give the operators flexibility. Ores may be divided into four broad classes as they are crushed, and then fed to either of the two milling sections in a reasonably constant mixture. This avoids frequent changes in milling rate, dilution, reagent dosage, and froth levels. The basis of classification of the ores may vary from month to month. Some ores contain considerable pyrite, others very litttle; if pyrite production is resumed this may become the criterion. Some ores contain almost no lead, while others contain 2.5 pct or more; at present it appears best to classify the ores according to lead content. Although none of the major ores now require grinding finer than 55-58 pct −200 mesh, some day a part of the feed may need much finer grinding than the balance. Then this factor may become the basic one.

Although equipped to handle oxidized lead ores, copper ores, or copper-zinc ores, up to now both sections have milled only lead-zinc ores typical of southwest New Mexico.

Most of the ores are replacement deposits in limestone. The nonsulphide gangue is chiefly composed of complex silicates of calcium and iron formed by hydrothermal metamorphism, with some quartz and magnetite. Pyrite and pyrrhotite are also important gangue minerals. Zinc is the chief value, followed in order by lead, silver, and copper. A recent month's operation of the mill is chosen to illustrate the various grades of ore. Nine ores totaling 19,180 tons were milled during the month with average grade of 1.5 ounces silver, 1.7 pct lead, and 11.9 pct zinc. The assays of the nine individual ores varied as follows: Silver 0.2 to 3.6 ounces, lead 0.2 to 6.2 pct, and zinc 6.5 to 16.7 pct.

Metallurgical control begins with laboratory tests. Samples received from prospective shippers are first crushed to ten mesh for flotation tests. Mill practice is followed as closely as possible to determine whether the ore will respond favorably to the established process. If it does, the shipper is given a provisional schedule. If the ore requires special treatment, the question of storing and then milling it separately is given careful study.

Sample rejects of all ores purchased are composited and tested at the end of each month. These



Elevation through the mill showing the relationship of the various sections from fine ore storage to concentrate loading. Concentrate goes from the flotation section to thickeners outside the mill building, and are returned to the filtering section shown in the elevation by two-stage pumping. Filter cake drops to the dock at right for loading into railroad cars.

flotation tests are used as a basis for calculating the financial outcome of individual ores.

Reagents: The reagents currently used in the mill are sodium cyanide, zinc sulphate, xanthate (Z-9), R-241, methyl isobutyl carbinol frother, Barrett 634, and hydrated lime. Consumption of these in November 1951 is shown in Table I.

Soluble reagents are dissolved in mixing tanks in a separate reagent plant and pumped to storage tanks on the mezzanine above the flotation floor. Water-insoluble liquid reagents are hand-pumped from drums to small storage tanks on the same floor. All reagents except lime are fed to the flotation circuits by Clarkson disc-and-cup feeders or Fisher and Porter valve controlled rotameter-type flowrators. Lime is brought to the mill in sacks and fed dry by Denver disc and Syntron vibrating feeders.

Sodium cyanide and zinc sulphate are fed to the mill scoop boxes, classifier overflows, and first lead cleaners. All other lead circuit reagents go to the head of the rougher bank. For zinc flotation, lime, xanthate, copper sulphate, and R-241 are fed to the conditioners, and copper sulphate, carbinol, R-241 and Barrett 634 are fed in stages to the rougher circuit. Lime is also added to the zinc cleaners.

Alkalinity: Lead flotation occurs at a natural pH of about 7.5, no attempt being made to alter it by addition of pH modifiers. For zinc flotation, however, the alkalinity is increased to pH 9 by lime additions, and for zinc cleaning is raised to pH 10.2 or 10.4 by further additions. Hourly pH determinations with a Beckman meter provide control. For the present there is no intention of going to automatic control.

General Considerations: The ores do not require fine grinding. Microscopic examination and assay of the various mesh sizes in the flotation tailing indicate very little loss of metal values in coarse middling grains (+100 mesh) when the ore is originally ground to 10-12 pct −65 mesh and 55-58 pct −200 mesh. Feed rate is registered by weightometers and manually controlled for pulp of this fineness.

Table I. Reagent Consumption November 1951
Pounds of Reagent Per Ton of Ore Milled

	Section 1		Section 2	
	Lead	Zinc	Lead	Zinc
Sodium Cyanide	.28	—	.43	—
Zinc Sulphate	.47	—	.44	—
Z-9	.02	.05	.01	.05
Carbinol	.05	.03	.05	.03
R-241	.05	.08	.04	.11
Copper Sulphate	—	1.81	—	1.60
Barrett 634	—	.03	—	.03
Lime	—	1.24	—	1.24

Classifier overflow density is held at about 45 pct solids. Automatic recording and control of this variable was attempted, and might have proved successful if only three or four ores were being milled, but with the present mixture of complex ores manual control is more satisfactory. Grinding circuit water comes from a constant-head tank and valve manipulation takes little of the operator's time.

For the most part the lead-zinc ores treated at the Deming mill are fast-floating and present no unusual metallurgical problems.

Metallurgical Results

Total recoveries of silver, lead, and zinc are in the 94-98 pct range, but recovery from individual ores varies widely. In the first eleven months of 1951 the mill received 38 different ores. Tests on these showed the following variations in total recoveries:

	Recovery
Silver	73 to 98 pct
Lead	55 to 97 pct
Zinc	67 to 98 pct

Power

Electrical power, purchased from the Public Service Co. of New Mexico, enters the mill substation at 13,800 volts, 60 cycles. Three transformers, 1500 KVA total capacity, reduce voltage to 480. An underground duct carries the conductors to a switch room which houses the disconnect switches for the four main mill circuits. Table II gives the power consumption for eleven months in 1951. The 250 hp synchronous motor on the Marcy ball mill and several capacitors installed in the crushing plant keep the power factor over 97 pct; in November 1951 the power factor for the entire mill was 100 pct.

Table II. Power Consumption 11 Months 1951

	Kilowatt Hours Per Ton Ore Milled
Crushing Plant	2.01
Grinding and Classification	9.37
Flotation	6.24
Product Pumping	.35
Tailing Disposal	.86
Thickening and Filtering	4.08
Lighting	.06
Experimental	.03
Water Supply	1.12
Assaying, Sampling	.18
Total	24.30

Water

Two Johnson deep-well pumps rated at 300 gpm and 26 ft head are connected direct to the mill water mains. A 6000-gallon elevated tank stores water for drinking and miscellaneous services, while a 60,000-gallon tank is kept full and cut into the mill mains automatically through a booster pump whenever pressure in the mains drops below 40 psi.

Auxiliary water pumps return overflow water from the lead concentrate thickener to the lead froth sprays, and from the zinc concentrate thickener to the zinc froth sprays. No other water is reused.

Miscellaneous

Materials are handled in the yard area and inside the larger buildings by trucks, Hough Payloaders, and a Wagner Scoopmobile, all of which find easy access tc any point.

The Du Pont color code is used throughout the mill: Red, indicates fire protective devices; blue, electrical controls; green, first aid cabinets; yellow with black stripes, stationary hazards; orange, mobile hazards such as cranes.

Direct daylight is excluded from the flotation section. Fluorescent lights are used to provide uniform illumination on all shifts.

Acknowledgments

Grateful acknowledgment is made of the help given the authors by the following: Mr. Bruce Matthews, assistant mill superintendent; Mr. B. Rickman, foreman; Mr. A. B. Romney, metallurgist; Mr. R. L. Salter, assistant on construction; Mr. Lou Jordan, engineer; and Mr. R. L. Marteeny, chief clerk.

Main bed phosphate is mined at the Gay mine for shipment to the Simplot Fertilizer Co. A Northwest Model 80D 2½ yd shovel and a Euclid end dump are used to do the job. Furnace grade phosphatic shale occurs in the bottom 25 ft of exposed pit face.

Western Phosphate Mining — A Growing Industry

by Charles W. Sweetwood

THE Western phosphate field, virtually ignored for 40 years, has been undergoing a rapid climb to economic importance. Until World War II there seemed to be no reason for developing the phosphate rock region of the Intermountain Empire. But the tremendously increased demand for food production and the needs of many branches of the chemical industry changed all that. Quite suddenly a report made by the U. S. Geological Survey at the turn of the century became important. The survey showed that 60 pct of the nation's phosphate reserves were in the western phosphate field.

Simplot Fertilizer Co. was one of the first to become actively interested in the development of the area. In August, 1945 the company began exploration of the Phosphoria Formation on the Fort Hall Indian Reservation. The result was the Gay Mine, about 18 miles east of Indian Agency Headquarters at Fort Hall, Idaho, developed to supply high grade phosphate rock for processing into superphosphate fertilizer at the company's Pocatello plant.

Six ft of high grade oolitic phosphate rock (ranging from 32.5 to 34.5 pct P_2O_5) were discovered at the base of the Phosphoria Formation by diamond drilling and dozer trenching. High quality and thickness, plus the fact that the formation lay flat and near the surface made strip and open pit mining possible. In one year of mining, starting June 1946, a total of 53,000 tons of main bed oolitic phosphate rock was hauled to Pocatello for processing into supersulphate. The first year's operation was in

CHARLES W. SWEETWOOD is superintendent of the Gay Mine of the Simplot Fertilizer Co.

Ridge C of the Gay mine produces phosphatic shale for shipment to the Westvaco Chemical div., Food Machinery & Chemical Corp.

reality large scale sampling and experimenting to determine the feasibility of economic mining. Development by core drilling continued and coupled with exposures by actual mining, brought to light the existence of some 21 ft of phosphatic shales above the main bed assaying at plus 24 pct P_2O_5. The quality was too low for acidualtion without prior beneficiation. Opportunity to use this shale came in 1947 and 1948 with fulfillment of several large export contracts. Blending the shale with high grade phosphate rock allowed a standard of 28 pct P_2O_5 to be maintained in compliance with contracts.

Westvaco Chemical Div., Food Machinery and Chemical Corp., became interested in the potentialities of these shales for production of elemental phosphorous in their proposed electric furnace operation. Experimentation with selective mining brought favorable results and actual mining stockpiling of "furnace grade" shales began in the fall of 1947.

With the mining of the shale for Westvaco, in addition to the recovery of high grade rock for Simplot, mining and loading facilities had to be expanded. Prior to 1948 trucks carried ore from the Gay mine to Pocatello, about 30 miles, or about 18 miles to the Union Pacific Railroad at Fort Hall. A standard gage railroad was completed in 1948 over the route from Fort Hall to the mine. The Idaho power company constructed a heavy duty power line to the mine. Other improvements at the mine, included dining hall, change room, repair shop, bunk houses and offices.

The stratigraphy of the Fort Hall Indian Reservation phosphate beds is similar to that found throughout southeastern Idaho. Generally, the higher phosphatic beds are found toward the base of the phosphatic shale member of the formation. Total thickness of the phosphatic shale formation is approximately 150 to 180 ft. It overlies the Wells formation and underlies the Rex Chert member of the phosphoria formation. The lower 65 ft of the shale member has consistently proved to be the "mineable" zone and is made up of the following:

The bottom 6 ft of oolitic phosphate rock, locally known as the "main bed," overlies the Wells Formation and is capped by 2 to 4 ft of limestone, referred to as the "cap rock"; this is followed by a relatively constant 25-ft thickness of siltstone, all of which is overlain by another 2 to 4 ft of limestone termed "false cap rock."

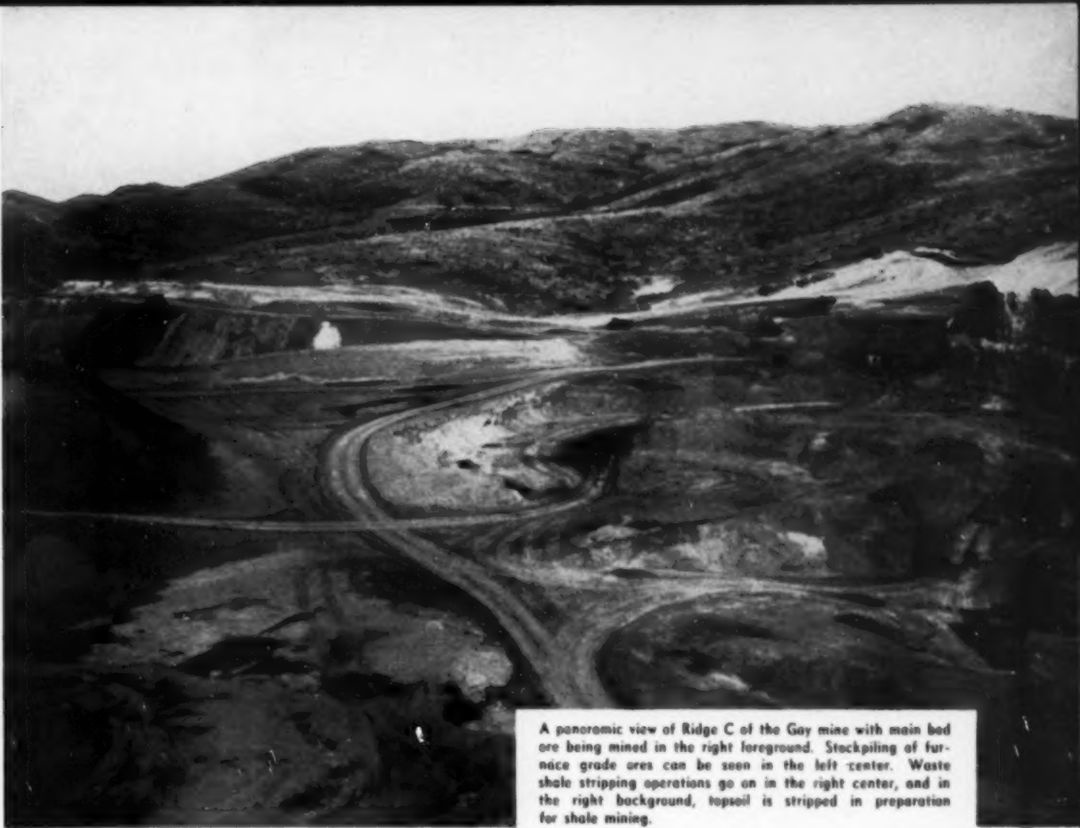
Immediately overlaying the false cap rock is a series of alternating phosphatic shales, limestones, and siltstones, all of which lens in and out but total 35 ft in thickness. Massive beds of light brown siltstones of very low P_2O_5 content follow. As progress is made upward in the section, the values of the phosphatic materials decline until the brown banded siltstones are reached. Several beds, or horizons above the brown banded siltstones are definitely phosphatic. They are, however, economically unfit for present day markets and operations.

Diamond drilling has shown the area pattern has a northeast-southwest strike with a 6 to 12-degree southeast dip. The dips are generally into the low rolling hills of the area, although several near dip slopes can be observed. The major faults and fault zones strike northeast-southwest and are always a primary consideration when laying out future mining panels. Constant trouble is encountered with operational faults, which have near vertical throws of 1 to 7 ft.

A panel is staked on the ground in accordance with geological data before stripping operations begin. Panel size is controlled by total tons of ore required during a given period of time, or by existing topographic or geologic conditions. Other factors

A Universal jaw crusher crushes all ore to —3 in. The unit samples all ore automatically and loads cars to capacity at the rate of 2000 short tons per eight working hours.

In the background on Ridge A topsoil stripping operations are carried on to uncover furnace grade shales. Waste shales are also removed by the Bucyrus-Erie Model 54B 2½ yd shovel and Euclid end dump trucks.



A panoramic view of Ridge C of the Gay mine with main bed ore being mined in the right foreground. Stockpiling of furnace grade ores can be seen in the left center. Waste shale stripping operations go on in the right center, and in the right background, topsoil is stripped in preparation for shale mining.

are fault blocks, depth of overburden and topography as it affects spoil area location.

The mining plan can be divided into four phases: Topsoil removal, waste shale removal, furnace grade shale mining, and the mining of the main beds of oolitic phosphate rock.

Topsoil removal is accomplished with the aid of three D-8 Caterpillar tractors equipped with Caterpillar Model 80 scrapers, with 15 yd capacity. The topsoil is of soft, powdery nature and the carryall is especially useful for the job. The total over-all loaded haul is designed never to exceed 500 ft for efficient operation. Down hill loaded hauls for the carryalls are usually possible because the phosphoria formation generally outcrops on slopes of gently rolling hills. The average thickness of topsoil is about 8 ft. Waste shales are removed by 80-D Northwest 2½ yd shovels and loaded into Euclid end dump trucks, Model 49-FD. Waste materials go into a dump no more than 1500 ft from the panel being stripped. The waste shale is placed in dumps separately and not contaminated with other materials.

Brown banded siltstones occurring above and below the waste shale zones are removed as separate units and placed in dumps apart from all possible quality materials. Model 80 Northwest 2½ yd shovels remove furnace grade shales, which are usually shipped immediately to Westvaco. A smaller Model 37-B Bucyrus Erie 1½ yd shovel is often used when greater selectivity is needed because of limestone lenses interbedded with the shales.

Close control of the shale is necessary because of its heterogeneous nature. Quality can vary rapidly in any section.

Main bed ore is removed by a 2½ yd shovel. Occasionally smaller shovels are required because of small operational faults that have dragged limestone and other foreign materials into the faulted zone. Main bed ore is never stockpiled. It is removed from the ground for immediate shipment.

All phosphatic shale and main bed ore is first moved by Euclid and Kenworth end dump trucks to the crushing and loading installation at the mine railhead. The haul is never more than one mile.

Ore is discharged into a 40-ton hopper and delivered by pan feeder to a jaw crusher, where it is crushed to −3 in. size. A 30-in. width conveyor belt carries the ore through an automatic sampler and directly into 65-ton hopper bottom railroad cars, rolled on to the scales while loading. An empty car is rolled into place by gravity to replace the filled car. Track space for 120 empty cars is available above the loading unit and for a like number of loaded cars below the loading unit.

The crushing unit was originally intended as a pilot plant for testing the efficiency of the automatic sampler and automatic railroad weighing scales. It soon became the main operating plant, capable of handling 2000 short tons, or an average of 30 cars per eight hr operating shift. Plans are underway for increasing output to 4000 tons per shift.

Time required to draw a caved stope in this Arizona copper mine was reduced to 11 months with conveyor belts. Using scraper loading in sub-drifts, the time necessary to exhaust the 200,000 tons in the block was 18 months. Lower maintenance costs are an important result of the installation.

CONVEYOR VS TRACK HAULAGE

If the problem is to get high production at lower costs, conveyor belt haulage may be the answer. Here is a cost comparison and story of operating units which may aid engineers in making the choice between troughed conveyors and railroad.

by R. U. Jackson

FACED with rising costs, the mining industry is looking to new methods and equipment to supply the relief necessary for profitable operation. Conveyor transportation is rapidly taking a superior position to motor tramming in underground mines because of lower installation, maintenance, and operating costs.

Haulage is always a critical problem in underground mining. The ease of operation and flexibility of a haulage system can have an important effect on ultimate mining cost. With the average tenor of ores on the downgrade, larger tonnages are required for the same metal recovery. This selection of a tramming system can be the key to a successful, paying plant. Because this shift is being made at an ever advancing rate, it may become necessary to tailor mining operations to the transportation system, rather than to select a haulage system fitting set mining methods.

In choosing a haulage system, the relative costs of the various methods are of the greatest importance. Adapting the mine to one or the other type of tramming is not considered in the cost analysis, and very likely would make little difference, outside of location of drifts and raises. In the cost sheet, the problem presents recommendations for a conveyor installation and a comparison of equipment, installation, operation, maintenance, and power costs are made. Under the track haulage estimate, no allowance was made for dumping equipment at the shaft pocket or bin. If this equipment was added to the balance sheet, the additional

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COST ANALYSIS

The Job:

Type of haulage: Main haulage
Distance: One mile
Grade: None
Capacity required: 300 tons per hr
Bulk weight: 80 lb per cu ft
Lump size: Small percentage over 24 in.
Operating time: 6 hr per shift, 2 shifts per day

CONVEYOR

Conveyor Equipment

2—36 in. x 2650 ft c/c mine conveyors, with internal tandem drive, automatic counterweighted takeup. Conveyor frame of 12-ft standard deck sections.

Heavy duty roller bearing troughing and return idlers.

Troughers spaced at 2 ft centers for 150 ft, at 3 ft for 350 ft, and 4 ft for the balance.

Belt 36 in., 4 ply, 42-oz duck or rayon carcass with 3/16 in. top cover and 1/16 in. bottom cover.

Drives 50 hp each.

Cost

Equipment	$226,000.00
Installation, 2000 man hr @ $2.00	4,000.00
Total equipment and installation cost	$230,000.00
Daily labor: 2 maintenance men @ 8 hr @ $2.00 and 2 supply men @ 8 hr @ $2.00	64.00*
Power 100 hp connected @ 2¢ kwh @ 16 hr	$ 32.00*

Recapitulation

	Per ton
Original investment 10 pct, $22,600.00	0.022
Interest @ 3 pct, $6780.00 year	0.008
Daily labor—$64.00 per 3600 tons	0.022
Power—$32.00 day per 3600 tons	0.012
Installation costs—$4000.00 for 9 million tons	0.000
Supplies @ 1 pct per year	0.002
Total cost per ton mile	$0.066

TRACK

Track Haulage Equipment

2—15-ton locomotives

120 mine cars

60 lb rail, thermit welded, and necessary trolley and feeder.

Cost

Equipment	$179,000.00
Installation, thermit welded track @ $3.50, trolley and feeder @ 20¢	32,500.00
Total equipment and installation cost	$211,500.00
Daily labor: 2 motormen, 2 brakemen, 1 maintenance man, and 1 track and wire man. 6 men @ hr per shift, 48 man hr per shift, 2 shifts, 96 man hr @ $2.00	192.00*
Power 0.2 kwh per ton @ 2¢	$ 14.40*

Recapitulation

	Per ton
Original investment 10 pct, $17,900.00	0.020
Interest @ 3 pct, $5370.00	0.006
Daily labor—$192.00 per 3600 tons	0.054
Power—$14.40 day per 3600 tons	0.004
Installation costs—$32,500.00 for 9 million tons	0.004
Supplies @ 1 pct per year	0.002
Total cost per ton mile	$0.090

Calculations based on 3600 tons per day, 250 days per year for 10 years, or 9.0 million tons.

* Per day.

Four million tons of 8-in. crushed dolomite was transported 7 miles for 4½c per ton mile at the Bull Shoals Dam, Ark. Shown is the main line conveyor to the secondary crushing plant. The belt is running toward the photographer. Note the section of conveyor that is operating on a steep downgrade.

Seven workers operated the Bull Shoals conveyor system that carried 650 tons per hr at 525 ft per min. After the belt was used for 2½ years, 75 pct of its useful life still remained.

cost would throw the balance more strongly to the conveyor. The balance sheet presented is hypothetical, since it would be difficult to obtain actual cases for comparison.

Careful examination of the cost analysis indicates that a conveyor installation would produce a net gain of $30,000 per year over a railroad. Operation and maintenance labor account for the majority of the savings.

One operation verifying this trend is an iron mine, producing over 500,000 tons of ore annually and employing about 90 men. The mine shows a transportation cost of $1.56 per thousand tons with belt conveyor against $5.46 with mine car haulage.

Underground Haulage Experience with Belt Conveyors

Some 56 years ago, Thomas Robins developed the three-pulley troughing idler. The rubber-belt conveyor then took its place as a leader in the transportation of pulverized, granular, and lumpy material in large quantities.

It has become an important mining transportation medium in copper, iron, and gold ores, bauxite, potash, limestone, and coal. The main differences in installation are based on lump size, unit bulk weight, abrasiveness, moisture content, and breakdown of sizes. These are factors considered when designing the conveyor system and does not mean that belts are applicable in *every* instance.

Looking along the main belt line from a transfer station. A total of 21 units were used to make up the seven miles of haulage.

Surface portion of a 4½-mile conveyor installation from underground to the preparation plant. To the right is the portal of the Ebensburg Coal Co.'s mine, where the 900-ft inclined belt comes to surface. The incline is fed by six underground conveyor units operating in tandem to make up three miles of main haulage.

Mine operators, however, do not seem to realize the extent and number of underground plants using this transport method, nor the years of experience back of belt transportation.

Because coal operators have been forced to search for the cheapest methods, conveyor haulage has its widest application in this field. In West Virginia, 195 mines are operating over 130 miles of belt; in Pennsylvania, 50 miles are in use; Kentucky has 45 miles. Today, coal mines in the U. S. are equipped with 300 miles of rubber-belt transportation and only 20 years ago, practically none existed underground.

To cite specific cases, two large coal mines in Illinois operate with all-belt haulage. One of these operations moves 1100 tons per hr of air-dox blasted coal, which contains a high percentage of plus 30 in. lumps. Some of these lumps weigh over a ton. The material is transferred from one belt to another without difficulty four times during the trip from the loading point to the preparation plant.

A Pennsylvania coal mine, in operation for over 40 years, was in an almost economically impossible situation. Four miles of track haulage were replaced with belt haulage, with a substantial saving.

In another instance a mining company now operating two belt equipped mines is installing all belt haulage with a capacity of 1200 tons per hr in a new operation. Equipment will include a slope conveyor with a 750 hp drive. Slope conveyors are definitely the medium for elevating bulk materials. In regard to slope conveyors vs. shafts, no coal mine has installed a shaft in the last eight years. However, in potash, copper, iron, and other ores local ground conditions such as intervening water tables, govern the selection of shaft hoisting.

Conveyor Belts in Metal Mines and Belt Life

Underground belt haulage is by no means limited to coal. Performance in metal and nonmetallic mines indicates that belt life is greater than was expected. Conveyors have had limited usage underground in metal mines, although no installation has been a failure. The limited use in many cases, may be a result of the mining system employed.

In Chile, a copper mine installed a 60-in. x 1412-ft conveyor to transport 5500 tons per hr of ½-in. ore. The belt is in operation after carrying 100 million tons of crushed material. Another copper plant in South America uses a 60-in. belt to move 3500 tons per hr of 12-in. copper ore with a belt life of 63 million tons. Belt life at a Rhodesian copper mine with a 7000-ft system averages 20 million tons.

In some cases the use of belt conveyors has advantages other than lower transportation costs. At the Miami Copper Co.'s mine in Arizona the installation of a conveyor belt in a sub-drift under the grizzly level has lowered the timber repair costs because of decreased time required to draw a block of ore. In a test block track haulage was replaced with a belt and 200,000 tons of copper ore in the stope was drawn in 11 months, compared to about 18 months which would have been required using track haulage.

The faster draw gave some relief to the timber sets which normally are subject to severe ground pressure, and also permitted timber repair without interrupting production. In situations such as this where any delay is costly the saving made with belt haulage is more significant than reduced cost. Once the mining is started in this type of block caving, the best results come with maximum production.

In the area of nonmetallics a Chilean nitrate mine with a 48-in. x 2000-ft conveyor transported some 80 million tons before replacing the belt.

What Is Necessary to Produce Results with Conveyor System

Good housekeeping and proper lubrication are the only ways to guarantee low cost with conveyor belts. Maintenance men must be properly instructed and given correct tools to carry out their jobs. It is important that they realize the responsibility involved and work with this in mind.

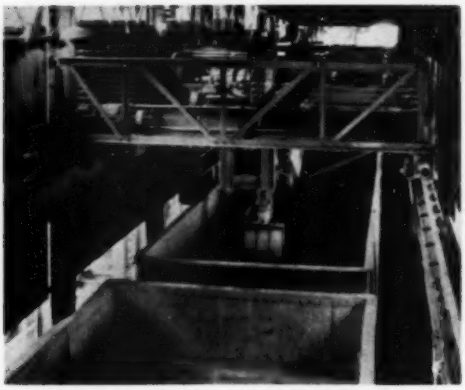
Belts must be inspected at frequent intervals and cuts temporarily patched until permanent repairs can be made. Idlers should be lubricated every 4000 hr of operation. However, good judgment must be used in determining the lubrication period. Wet or dusty conditions can radically alter requirements, too. It must be borne in mind, however, that too much grease is harmful. To be sure, an idler can be pulled out, taken apart, and examined visually. This is a small cost to assure proper lubrication. Machinery maintenance can be carried out in the same logical manner and low haulage cost will invariably follow.

One of the intangibles of a belt installation is the safety feature. Mine records show that both lost time and fatal accidents due to belt haulage have been remarkably low.

Continued study and more improvements are necessary to lower transportation costs underground. Information has been given showing the advancement in the use of belt conveyor haulage systems and to indicate their efficiency and low cost operation with reduced manpower requirements. Cases have been cited showing the true life expectancy of conveyor belts and conveyor machinery.

Four Solutions To:

Cold Weather Unloading



The Kinney car unloader at work at the Detroit Edison Co. plant. Frozen coal is removed from cars without the use of heat.

Broken material, whether it be iron ore, coal, or the copper porphrys follows a similar pattern when frozen, i. e., it will not dump. Coal operators handle large quantities of material in cold, damp weather, and have had to find a solution to the problem. But their methods should work as well, regardless of the ore in the cars. Any operator that must leave loaded cars out in freezing weather can find something to help him.

HARD as it is at any time to provide more than three million tons of coal each year for power plants, the task really gets rugged in the wintertime when the fuel freezes solid in the cars.

At the Marysville plant of the Detroit Edison Co. the problem of unloading cars of frozen coal was severe because there was an unloader over only one of the two tracks in the coal unloading house. Moving downward by their own weight, with no driving force to aid them, the spuds of this old unloader were unable to break through the crust on a frozen coal car.

20,000 Lb Force

To meet this problem, a new mechanical car unloader was installed over No. 2 track and placed in service in December 1947. It is a Kenney type unloader built by Heyl & Patterson, Inc. of Pittsburgh, Pa., a modification of equipment originally designed to unload iron ore cars. The major advantage of this unloader is the force with which it can drive the sharp spade on the end of its ram down through the pockets of a frozen coal car.

The unloader consists of the following major units: 1. A traveling bridge structure. 2. A traveling trolley. 3. A retractable and oscillating ram. 4. The operators cab and controls.

The bridge is provided with safety hooks to prevent its being lifted from the rails by the upwards push of the ram. The bridge will travel 190 fpm and has sufficient traction on its four wheels to move three loaded 70 ton cars.

The trolley runs on tracks supported by the bridge girders and also has upper tracks to control the push from the ram. Two of the single-flange trolley wheels are driven by a one and a quarter hp dc motors equipped with magnetic brakes.

The ram is raised and lowered by a 20 hp dc motor equipped with a magnetic brake. A full stroke of a little over 14 ft is given the ram through its rack and pinion drive and the spade can be pushed down into a loaded car with a force of about 20,000 lb.

A 15 hp dc motor with a magnetic brake is applied to this drive and is able to swing the ram 45° either side of the vertical.

The usual procedure in emptying a coal car is first to drive the sharp spade down through the coal until the pocket is opened and the coal starts to run out the hopper. When all the hoppers are open, the sweeping action of the ram and spade is used to move the remainder of the coal in the car over to the open hoppers. The travel of the trolley permits cleaning the sides of a coal car with the edges of the spade, using the sweeping motion of the ram. It is still necessary to clean out the corners of the car by hand.

Final Clean-Out

This unloader was a new development and there were improvements to be made before it was completely satisfactory, such as reduction in speed and conversion to ac drive motors. However, it helped at Marysville and unloaded many frozen cars that would have taken a lot of heating to thaw out enough for the old unloader to handle.

The mechanical coal unloader described above was a considerable improvement but still left a job to be done by hand—the final cleaning out of the coal cars. The ram had the ability to open a hole through the hopper cars, but did not meet required unloading speed.

To eliminate hand cleaning a car shakeout, manufactured by Hewitt Robins, Inc. was added to the installation. This equipment is used to clean the coal car out after the unloader has provided an

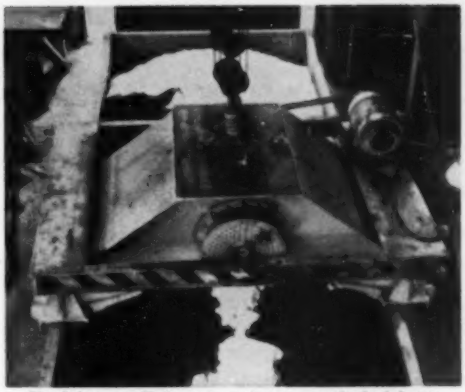
opening through the coal. The main parts include a power winch for raising and lowering the shakeout, and a trolley to position the shakeout over the coal cars.

The car shakeout itself is an all-welded "U" shaped steel frame that rests on the top frame of a car. The shaking mechanism contained in the frame consists of a motor-driven eccentric shaft. On either end of the shaft is a counterweight with removable plates for adjusting the intensity of the shaking section. The 20 hp motor that drives the eccentric shaft is mounted on a spring-suspended frame.

The trolley, built by Heyl Patterson, Inc., is also of all-welded steel construction. It is driven by a two-horsepower motor and uses the same rails as the coal car unloader. Its speed is approximately 114 fpm. The winch for hoisting the car shakeout is located in the center of the trolley. The winch drum is powered through reduction gears by a 6 hp motor.

The operations of the car shakeout, hoisting winch, and the trolley are all remotely controlled from the operator's cab on the coal car unloader. The sequence of operations is as follows: the car to be unloaded is spotted over the unloading pit and opened, the car shakeout is positioned over the center of the car and lowered onto the top frame of the car, sufficient slack is left in the hoist cables so the shaking action is not transmitted to the trolley. The shakeout is then put into operation until the car is clean. Usually it takes only a minute or two to clean out a car.

To date, the car shakeout has performed very well. Climbing into the cars to clean them by hand has been completely eliminated. Unloading time and effort has been reduced. The use of this machine should greatly facilitate the uninterrupted flow of coal to the power house coal bunkers.



The Robins car Shakeout in action on a carload of frozen coal at Detroit Edison. It has helped to solve a serious winter problem for the company.

The combination of units described above approaches a workable means of unloading frozen cars without heat. The use of heat, however, has not been eliminated in our operations. Heat applied to the car hoppers either by hand torches or by track thawing pits reduces the time required for unloading even with the improved equipment described above.

Material for this article was obtained from "Coal to Kilowatt," Detroit Edison publication.

Car Shakers for Unloading Frozen Coal

by T. S. Abbott

THE difficulties which arise when unloading wet or frozen coal from hopper cars can have far reaching effects on the industry involved. If coal cannot be unloaded as fast as is required, generating capacity and process steam production is limited. This, in turn, limits productive capacity.

In manual unloading, a pronounced danger to personnel exists. Unloading by hand may also damage the cars.

There are several ways for minimizing the inconvenience of unloading wet or frozen coal.

Hopper cars can be redesigned so that they are truly self-cleaning. The average car in use today has hopper slope angles of approximately 30°. This angle is less than the angle of repose of the material in the car, in most cases. Therefore, something besides gravity must be used to move the material to the hopper openings. There are thousands of cars in the United States carrying sand, coal, gravel and other bulk materials. The railroad manufacturers estimate the life of a car at twenty years. Therefore, even if immediate changes were made, it would be a long time before redesign of cars would help.

The time for unloading frozen coal may be four to eight times longer than for unfrozen coal. The frozen coal may be a thin crust on the outside surfaces or 2 ft of solidly frozen material. Some operators have reported receiving cars completely frozen solid. In the northern states difficult unloading conditions can be expected for about six months. Even though coal may be thoroughly dewatered or heat dried before loading, rain and freezing temperatures in transit can cause unloading difficulties.

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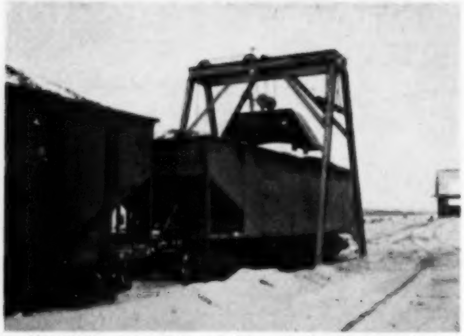
The majority of coal preparation plants arrange to clean the cars before loading, but in some cases not all the snow is removed. Unloading can be simplified if transit time is reduced as it takes several days for deep freezing to occur. Unloading conditions can be improved if the per cent of ⅛-in. x 0 coal is less than 15 pct in the ¾ in. x 0 or 1½ in. x 0 size.

Mechanical Unloading

The preferred means of unloading large quantities of coal is a rotary car dump, but an installation costs in excess of $100,000. A second most desirable method is the use of one or more car shakers. A car shaker installation would cost approximately $10,000. These approximate estimates do not include hoppers, conveyors, or building structure.



Allis-Chalmers car shaker used in unloading frozen coal at power plant of the Lake Superior district Power Co., Ashland, Wis.

A car shaker will unload approximately four to eight cars of coal per hr depending upon size of the hopper, conveyor facilities and switching time. Small car shaker installations can be operated by one man or possibly two if assistance is required for opening and closing car doors. The car shaker eliminates the necessity for personnel to enter the car to pick or shovel the coal to start it flowing out the doors.

Because manual unloading of cars is a hazardous job, it is sometimes difficult to obtain personnel to do this work. Installation of mechanical equipment will go a long way toward raising unloading crew morale. With proper design, a car shaker will give years of low maintenance service and unload coal or other material without damaging cars.

How a car shaker is installed depends to a large extent upon where the coal must go from the unloading station. A single hopper unloading station is the simplest—where the car is always spotted at the same place for unloading. The car shaker is then located over the hopper and near a permanently mounted hoist. With this type of installation the hopper beneath the tracks should be large enough for at least one-half of the car capacity; unloading operations will be quicker because one or two runs per car are more effective than many short runs.

A multiple track hopper installation can be made. Several cars are spotted on one track or on parallel tracks and a hoist mounted on a monorail to move the car shaker from car to car for unloading.

Car shaker duty is severe. Large commercial units today weigh five tons, and must develop sufficient energy to transmit motion to a sixteen ton car containing fifty or more tons of material. Car shakers act on the spring supported car, vibrating it sufficiently so that the bottom sloping plates will feed the coal to the hopper doors.

Too low a vibration speed will not produce relative movement between the bottom sloping plates and the material. Too high a speed will cause car failure.

Car Shaker Limitations

Car shakers cannot be used for unloading where coal is frozen solid. For such conditions, auxiliary thawing equipment must be used; thawing pits or hand torches.

If the car is not frozen for more than 2 ft all around, the shaker will break up the frozen mass, but difficulty may be encountered in passing large chunks through the car discharge opening and the track stationary grate. Thawing equipment is recommended for such conditions.

Freezeproofing Coal with Calcium Chloride

by William E. Dickinson

MODERN washing practice has added to the problems of shipping and handling coal in northern states during the winter. Even dried coal shipped in the winter can become frozen enroute, if exposed to freezing rain.

These remarks are confined to a method of preventing frozen shipments by freezeproofing with calcium chloride. By freezeproofing we actually mean that the coal can be unloaded, even though containing some frozen lumps.

For many years, coal has been both freezeproofed and dustproofed by treatment with calcium chloride. It is ideally adapted for this protection, because it is low in cost, clean and odorless. When properly used, it will freezeproof coal at lowest winter temperatures. Furthermore, calcium chloride treatment does not leave a deposit or otherwise change the appearance of the coal, and its use presents no fire hazard.

Method of Treatment

Snow and ice should be removed from cars before loading.

The choice between flake or solution treatment is determined by surface moisture content of the coal. A simple rule-of-thumb method is: Use flake if the coal contains enough surface moisture to cause some drainage from the cars after loading, if no drainage, solution is indicated. Experience indicates solution may be more effective than flake in maintaining drainage.

W. E. DICKINSON is with the Calcium Chloride Institute, Washington, D. C.

Quantities

The actual quantity of calcium chloride required to freezeproof a shipment of coal depends upon: (1) Surface moisture on the coal. (2) The size consist of the coal. (3) The lowest anticipated temperature to which coal will be subjected. (4) The amount of precipitation while coal is in transit.

Recommended quantities for various temperatures and percentages of surface moisture are the minimums required to insure that the car of coal can be unloaded at destination without delay.

Quantities Recommended for Freezeproofing Coal Shipments

Temperature °F	Lb Flake Calcium Chloride Per Ton of Coal: 3 Pct Surface Moisture	6 Pct Surface Moisture	9 Pct Surface Moisture
+32 to +15	3.0 to 4.5	6.0 to 9.0	9.0 to 13.5
+15 to 0	4.5 to 6.0	9.0 to 12.0	13.5 to 18.0
0 to −15	6.0 to 7.5	12.0 to 15.0	18.0 to 22.5

Note: As the proportion of fine coal in the shipment increases, use the larger amount indicated in the table for the conditions stated.

Flake Calcium Chloride

Flake should be uniformly applied to coal using vibrating feeders or other mechanical devices and should be applied at some convenient point prior to loading such as the loading boom, chute or conveyor. If mechanical devices are not available, the flakes may be added manually as the cars are being loaded.

Washed coal should be dewatered as much as possible before calcium chloride treatment and when mixing washed and dry coal, the flake calcium chloride should be applied on the wet coal before combining the two.

Calcium Chloride in Solution

A 32 pct calcium chloride solution (4½ lb of flake calcium chloride per gal) is recommended. This solution should be sprayed uniformly on the coal as it comes off the boom or loading chute. Shielded nozzles should direct all spray on the coal.

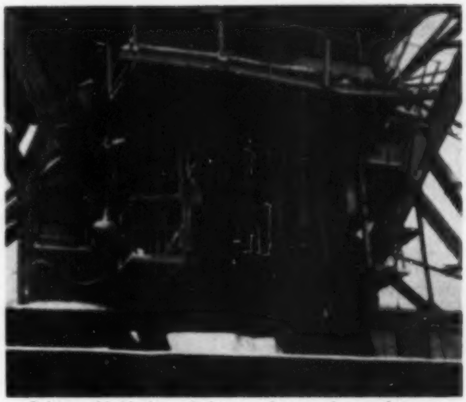
To determine the gallons of 32 pct solution required per ton of coal, divide the quantity indicated in the above table by 4½. For example, to apply the equivalent of 6 lb flake calcium chloride per ton:

$$\frac{6}{4.5} = 1.33 \text{ gal solution per ton of coal.}$$

Equipment

Tanks: A metal tank in which to dissolve the flake calcium chloride is recommended. A concrete tank can be used if properly constructed. A wooden tank is satisfactory if thoroughly coated inside with asphaltum paint and constructed with heavy adjustable hoops. Calcium chloride tends to shrink wood. Calcium chloride can also be had in tank cars or by truck delivery.

It is recommended that the tank be equipped with a suitable ⅛ in. wire mesh basket suspended above the bottom of the tank into which the flake is poured. The tank should also be equipped with an agitator or pump for thoroughly mixing the solution. By arranging piping with an additional valve or two, the same pump that supplies the spray nozzles may be used for mixing the solution by recirculation.



Calcium chloride is applied to ore by use of spray bars, as the material is loaded into cars. Each car receives a gallon of solution for each ton of ore loaded.

A screen should be used over the tank opening to catch paper or other foreign material and a removable screen should be placed in the tank outlet line, the mesh being small enough to prevent clogging of the spray nozzles.

A 32 pct solution of calcium chloride weighs approximately 11 lb per gal. Water weighs 8 1/3 lb per gal. This difference in weight should be taken into account when constructing tank supports.

Pumps: Any centrifugal pump that will deliver the required volume of solution at from 5 to not over 40 psi pressure at the nozzles, is recommended.

Piping: Iron or steel pipe is satisfactory for use with calcium chloride.

Meters: It is recommended that a meter be installed in the line for control of quantities. An oscillating piston type meter is preferred. Ordinary water meters can be used if cleaned periodically.

Unloading Frozen Coal with Oil Torches

by Roger E. Curfman

UNLOADING a frozen coal car can be dangerous, costly, and time consuming. Industrial coal users with operations in cold winter areas must keep these three factors constantly in mind. The Cleveland Electric Illuminating Co. operates plants in northern Ohio on Lake Erie, where temperatures during winter months average about 30°F. Enough coal must be on hand to keep operations running smoothly at three generating plants, where 150 cars are unloaded daily.

Oil torches are used to thaw coal and thus far it appears to be the best method. It fulfills the safety factor, but only if precautions are taken. The cost factor has been disadvantageous. Eight men are needed to do the job, with several torches going on all sides of the car. Overtime pay makes it relatively expensive and supervision is constantly required. If the coal is frozen in a block, the torches serve only to release the block from the car. The possibility of a 60 ton battering ram smashing into dumping equipment when the car is emptied, is ever present. In addition, supervision is needed to prevent damage to the car itself.

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The average cost per ton, over the normal dumping charge, is about \$0.09¾. When broken down it looks like this:

Fuel oil for torches	1.87¢
Labor for torches	2.52¢
Labor for blending storage coal	0.48¢
Overtime Labor	4.88¢
Car Damage	0.03¢
	9.78¢

Unloading speed at the plants is not important. At one installation four steam lances are driven into the car and badly frozen coal will thaw in about one half hour. The car shakeout easily removes the coal after steaming. The car must be unloaded immediately after it has been steamed or it will refreeze tighter than before. The method is time consuming but labor costs are relatively low and the safety problem not as serious as in the oil torch method.

Books for Engineers

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

The Mining Journal Annual Review. The Mining Journal, London. $1.00, 200 pp., 1952.—Contains 70 specially contributed articles reporting on events in the metalliferous mining industry throughout the world during 1951. The articles are written from the standpoints of: Economics, world mining fields, progress reports on Empire mining companies, and British mining taxation. The review has appeared for a period of 20 years. Technical developments in mining, ore treatment, refining, and metallurgy are covered.

Dun & Bradstreet Reference Book. Dun & Bradstreet Inc. 4000 pp., 1952. —Changes, brought about by the expanding economy, have been made for the three million listings for the United States and Canada. Planning for the changes started more than 10 years ago. New listings have been included for the metals and mining industries on all levels. The rapid development and specialization of industry have made the old symbols used in previous editions outmoded and a new system is substituted.

Atlas of World Mineral Resources, Edited by William Van Royen and Dr. Oliver Bowles. Prentice-Hall Inc. $10.75, 180 pp., 1952.—Scheduled for publication this fall, the book contains information that has never been published in map form before, as well as much that has not been presented in map form for many years. Facts on 29 minerals most important in world trade are dealt with, along with interpretation of all data available on each mineral, discussion of the world's mineral economy from viewpoint of specific minerals related to mineral exploration, geographical distribution of world's consumption, and appraisal of the adequacy of world's mineral resources.

Electric Analog Computers, by Granino A. Korn and Theresa M. Korn. McGraw-Hill Book Co. $7.00, 378 pp., 1952.—The book contains practical setup procedures and the application of dc analog computers to representative practical problems. The theory, design and operation of linear computing elements amplifier circuits and control circuits are discussed. Analog multiplication and function generation developments are covered. Design and construction of complete computer installations are presented for various uses in industry.

Controllers For Electric Motors, by Henry Duvall James and Lewis Edwin Markle. McGraw-Hill Book Co. $7.00, 426 pp., 1952.—The book contains descriptions of the construction, performance and operation of all types of commercial motor controls. Details are given on protective devices, with brief instructions for maintenance and installation. Special control systems, as those used in elevators and steel mills are omitted. The book is aimed at technical students, operating engineers, and application engineers. A limited knowledge of electric motors is needed.

Building and Civil Engineering Plant, by Spence Geddes. Crosby Lockwood & Sons, Ltd., London. 30s, 302 pp., 1951.—The book attempts to supply a comprehensive reference on building and civil engineering equipment, making possible considered purchasing and efficient application. Practical and economic considerations are discussed in the opening chapters. Succeeding chapters describe equipment, classified by work types; covering constructional features, application, operation, output and labor needed.

Statistical Theory With Engineering Applications, and **Statistical Tables and Formulas,** by A. Hald. John Wiley and Sons, Inc. $9.00 and $2.50. 783 pp. and 97 pp., 1952.—It was the author's desire, in writing the book, to cover a large part of the more practical theories developed in the past 50 years. Although written largely from a mathematical standpoint, each important theorem is accompanied by examples derived from practical experience. Main stress is on the normal distribution and the significant tests connected with it. The smaller volume contains a comprehensive set of statistical formulas and tables. Notes on additional material are included.

Surveyor's Field Note Forms, by Clarence E. Bardsley and Ernest W. Carlton. International Textbook Co. $3.00, 120 pp., 1952.—The book provides a sample set of field notes for use with classroom lectures and as a supplement to a standard text on plane surveying. It is intended especially for the student who is just beginning the subject. Introductory suggestions are included, and emphasis is placed on careful, systematic note taking for future reference.

Electronic Measurements, by Frederick Emmons Terman and Joseph May Pettit. McGraw-Hill Book Co. $10.00, 707 pp., 1952.—The book treats the measuring problems often encountered by radio or electronics engineers. It is intended to serve the practical engineer as a compendium of measuring techniques and equipment and the student as a general textbook on measuring principles. The book is essentially a revision of *Measurements in Radio Engineering,* but the present volume is double in size and covers a wider range.

Fuels and Combustion, by Marion L. Smith and Karl W. Stinson. McGraw-Hill Book Co. $6.50, 340 pp., 1952.—The authors present a concise but comprehensive treatment of solid, liquid, and gaseous fuels, the fundamentals of the combustion process, and the application of combustion principles in furnaces, oil and gas burners, internal combustion engines, gas turbines, and rockets. Problems and literature references accompany each chapter. The chemistry of combustion is understandable by students with only elementary backgrounds.

Graphic Methods For Solving Problems, by Frank A. Heacock. Princeton University, $1.90, 113 pp., 1952.—The book is a reading guide to current literature on graphic methods applied to solution of technical problems in the fields of engineering, pure science, and industry. Explanatory text precedes the bibliography in each of eight sections: Simple graphs and charts, the hydrograph, geometric diagrams, network charts, vectors and mechanics, descriptive geometry, nomographs, and graphic analysis. More than 600 references are given, with brief abstracts.

Die Wassererschliessung, by Dr. W. Classen. Sponsored by Deutscher Verein Von Gasund Wasserfachmannern. DM 68, 421 pp., 1952.—The finding and developing of underground water supply is divided into two parts. The first part deals with general geohydrology, ground water chemistry, well sinking and the installation of equipment, the adaptation of spring flows, pump calculations, and estimation of yield. The second part covers geoelectrical prospecting methods, including limitations and range of applications, and discussion of the electrical characteristics of water-bearing ground. Both sections are separately indexed and contain bibliographies. A concluding section contains short technical contributions by consulting firms and equipment manufacturers.

Steam Power Plants, by Alexander H. Zerban and Edwin P. Nye. International Textbook Co. $7.50, 524 pp., 1952.—The book is a comprehensive treatment of mechanical engineering fundamentals of steam power plants. Special attention is given to the matter of heat transfer in power plant processes. The reasoning for the latest steam generator design is explained with regard to heat transfer requirements. The book includes meters and controls and the relation of all phases of power plant operation to economic considerations, with examples and problem solutions.

Glass and Chemical Sand Manufacture In the Edwards Paddle Scrubber

by Will Mitchell, Jr., T. G. Kirkland, and R. C. Edwards

A scrubber of new design has been invented for the beneficiation of glass and chemical sands. The machine is described and its capacity and metallurgical efficiency compared with the performance of other sand cleaning devices.

THREE years ago, when the Process Research Laboratory at Allis-Chalmers Manufacturing Co. sought a remedy for the increasing cost of disposing of great quantities of spent sands from foundries, R. C. Edwards developed a continuous inclined paddle scrubber which removed a substantial amount of the charred coatings from the grains with a surprisingly small amount of degradation of the sand itself.[1] The satisfactory results obtained on foundry sand, using this piece of equipment in conjunction with classification, suggested that a similar procedure could probably be applied to the cleaning of other sands, such as those required by the glass and chemical industries.

This paper describes the flowsheet and the results of tests performed on impure natural sands. The four materials tested were silica sands obtained from widely separated geographical areas. Three of the sands tested contained various amounts of iron oxide and the fourth an excessive amount of calcareous bonding material. The iron oxide was present predominantly in the form of grain coatings, which were scuffed by the paddle scrubber, and in all cases a product was manufactured which would meet one of the sets of specifications imposed by the glass industry.[2]

The Edwards paddle scrubber used in these tests consists of a rubber-lined steel cylinder 18 in. in diam and 48 in. long, mounted on a frame so that its slope can be varied 5° to 35° from horizontal. This variable slope permits controlled flow of material through the machine. Rubber paddles rotate inside the cylinder and are secured to a center shaft 2 in. square, which extends through the entire length and is turned at both ends so that it can be supported by sleeve bearings centrally fixed in the end plates of the shell. At the upper or feed end it extends through the bearing on the end plate into a sheave, to which the motor drive is connected by means of Texrope transmission.

The four sets of paddles mounted on the shaft, two in each set, are made of rubber belting, ¼ in. thick, 9 in. wide, and 16 in. long. The two paddles in each set are secured on opposite sides of the square shaft, but in the same longitudinal position along the shaft axis. In other words, the two paddles in the set operate 180° apart. Each successive set is mounted 90° from the preceding one. One end of each paddle is bolted to the square shaft and the other weighted by a steel bar, 1x1½x9 in. long, which is attached about ½ in. from the outer tip of the paddle. By centrifugal force at proper shaft speed, these weights force the paddle surface to rub smoothly against the rubber lining of the cylinder. During operation, the sand forces its way between the paddle face and the surface of the cylinder, providing the necessary abrading action as the pulp flows by gravity down the length of the cylinder.

The capacity of the machine is limited by the amount and the hardness of the coating material and by the quality of product desired. The degree of scrubbing is governed by the retention time in the scrubber, which in turn may be changed by varying the feed rate, water content of the feed, and

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The Transactions papers appearing in Mining Engineering during 1952 will subsequently appear in AIME Transactions Vol. 193, and may be permanently referenced to that volume.

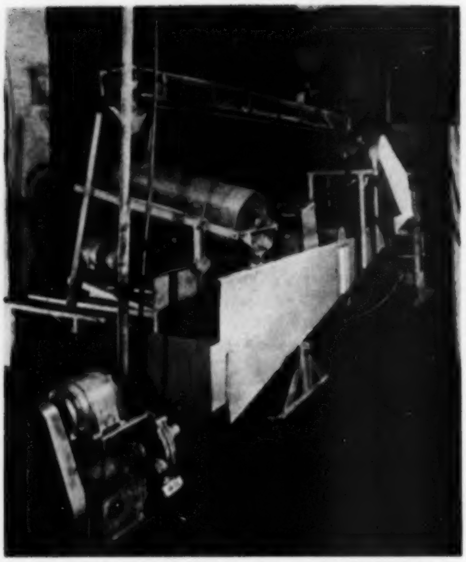


Fig. 1—Feeder, scrubber, and classifier circuit in operation.

slope of the tube, as well as the speed of the paddles. In the tests to be described the shaft was turned at 400 rpm, the slope of the tube was maintained at 13°, and the water content of the feed was kept at 18 to 20 pct.

The wear of the rubber lining of the cylinder is negligible if proper conditions are maintained, that is, regulation of feed rate and slurry so that the bulk of the sand grains scrape against each other rather than against the rubber surfaces. Seventy-five tons of foundry sand have been cleaned in a continuous process without causing measurable wear of the lining. Until recently the paddles had been fashioned from neoprene-covered belting, but because of an excessively high rate of wear they now are being manufactured from abrasion-resistant natural rubber. Data on the life of the new paddles are not yet available.

Table I. Iron Oxide Distribution

	Distribution, Pct	Weight of Total Sample, Pct
Fe_2O_3 in −200 mesh cementing material	25.0	0.06
Fe_2O_3 as stain on quartz grains	45.0	0.11
Fe_2O_3 in quartz grains	15.0	0.04
Fe_2O_3 as partially decomposed magnetite	8.0	0.02
Fe_2O_3, as partially decomposed pyrite	4.0	0.01
Fe_2O_3 as free limonite	3.0	0.007
Total	100.0	0.247

Pilot Plant Tests

The general flowsheet for tests of the paddle scrubber included a coarse grinding device, required only if the particle size was larger than 20 mesh, a vibrating screen, the paddle scrubber, a rake classifier operated to separate at 200 mesh, a thickener, and a filter for dewatering slimes. A rotary drier was used for dewatering the cleaned sands. Fig. 1 shows the feeder, scrubber, and classifier unit in operation. A schematic flowsheet of the process is shown in Fig. 2.

In wet screening, 12 to 18 pct moisture was removed from the sand by means of a rake classifier where the −200 mesh fraction was eliminated. In dry screening, it was necessary to add water by means of a spray at the feed end of the scrubber; the fines were allowed to pass through the scrubber to be removed later in the final classifier.

The four sands used for this series, obtained from Nevada, Michigan, Illinois, and Ohio, possessed two properties in common: the main impurity to be eliminated occurred predominantly as 1—coating on the grain surfaces or, 2—as cementing material binding the grains together. Thus all could be beneficiated by a scuffing action and were treated in general as described in the flowsheet.

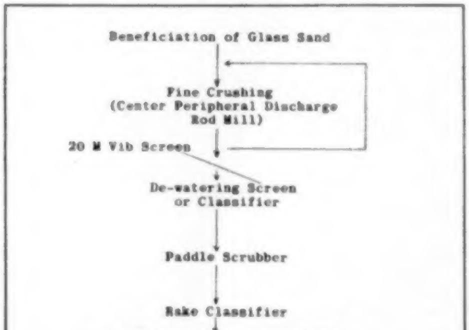
The Nevada Silica Sand Test: The raw material for this test came from a deposit near Overton, Nevada, and was treated to determine whether or not the iron content could be lowered sufficiently to make it acceptable as glass sand. It was a light brown, loosely cemented, quartzitic sandstone. Particle size varied from cemented pieces as large as 4 in. in diam to individual grains smaller than 74 microns. Cementing material consisted of siliceous clay minerals, quartz, chalcedony, opal, and iron oxide.

The quartz grains were predominantly equidimensional; however, their surface was not smooth. The grain size of the quartz varied from 74 to 840 microns; the average was 210 microns.

Iron oxide occurred as free limonite, limonite stain or coating on the quartz grains, and limonite and magnetite contained within the quartz grains. The Fe_2O_3 content in the sample as received was 0.247 pct, the average of six analyses. Table I shows the distribution of the iron minerals, the form in which they occurred, and the percentages of each.

The feed rate to the scrubber was 1000 lb per hr, and the power requirement was 13 kw-hr per ton. However, the scrubber was not operated at capacity for this test.

Fig. 2—Flowsheet of cleaning process.



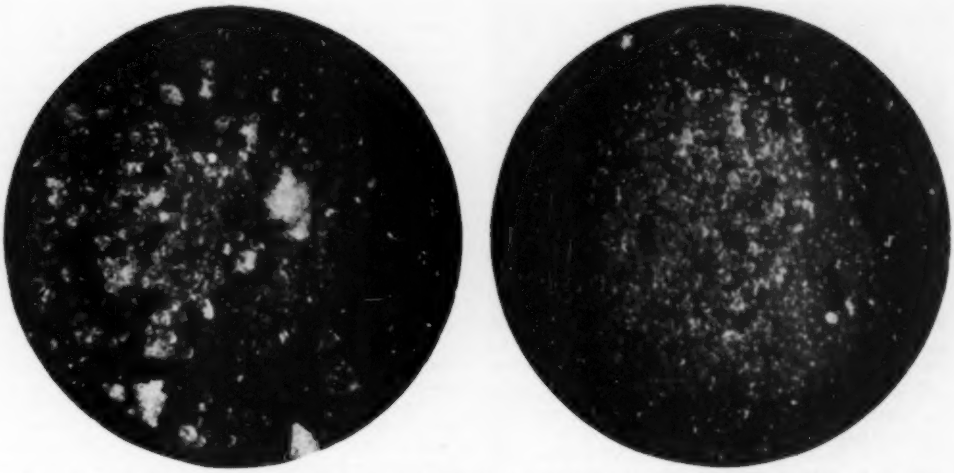


Fig. 3—Photograph of the feed (left) and the classifier sands (right). Note the clusters of cemented quartz crystals in the feed and the absence of these clusters in the classifier sands. X4.

Complete chemical analyses of the feed and the deslimed sand scrubber product are shown in Table II. A metallurgical balance for the classifier products is shown in Table VII. Observation by means of a microscope showed that the iron oxide remaining in the product did not occur in the form of grain coatings, but as inclusions in the grains and as precipitated oxide in the internal cracks. The surfaces were well cleaned. Reduction of impurities in the Nevada sand accomplished by the paddle scrubber and classification without further processing make this sand acceptable as "fifth quality glass sand," used chiefly for sheet glass.

The Michigan Silica Sand Test: The sand for this test, found in Wayne County, Mich., was to be used in the chemical industry. Impurity elimination of lime and magnesia was of prime importance, with iron oxide elimination of secondary importance. The sand as received was almost pure white in color, having the general appearance of beach sand, and was −20 mesh in size. The quartz occurred as clear, well-rounded grains, approximately 1/10 of 1 pct of the grains coated with limonite stain. A very small amount of pyrite was noted in the otherwise clear crystals. Calcite and dolomite, 1 pct each in quantity, were present in two forms, as individual grains and as cementing material bonding together grains of quartz. Small clusters of quartz grains surrounded a grain of calcite, and often the whole cluster was cemented together by calcium carbonate. Small pieces of calcite adhered to some quartz grains, indicating that during the mining and previous milling operation a number of clusters of cemented quartz grains had been broken.

The scrubber tests on the Michigan sand were conducted at three different feed rates, 500 lb per hr, 1000 lb per hr, and 1500 lb per hr. Chemical analyses of the feed and products from the three tests, with the calculated percentage elimination of contaminants, are shown in Table III.

Table II. Chemical Analysis of Feed and Sand Scrubber Product

	SiO_2 Pct	Fe_2O_3 Pct	Al_2O_3 Pct	CaO Pct	MgO Pct	TiO_2 Pct	Loss on Ignition
Feed	95.09	0.247	2.190	1.08	0.292	0.032	0.56
Classified Sand scrubber Product	97.90	0.061	1.27	0.292	0.141	0.016	0.20

Observation of the finished product under the microscope showed that the action in the sand scrubber circuit had broken the clusters of cemented quartz and had dislodged and eliminated the calcite adhering to the quartz grains. Fig. 3 shows the feed and also a classifier sand product. The feed shows clusters of quartz crystals cemented together with calcite. As can be seen in the picture of the product, the clusters are completely broken up and the quartz crystals are clear. Much of the iron stain was removed from the quartz grains, but the included pyrite remained.

The power requirement of the scrubber was as follows: 500 lb per hr feed rate, 20 kw-hr per ton; 1000 lb per hr feed rate, 12 kw-hr per ton; 1500 lb per hr feed rate, 9 kw-hr per ton.

The MgO and CaO content of the Michigan sand

Table III. Chemical Analyses of the Feed and Paddle Scrubber Products, Michigan Sand

	Loss on Ignition	SiO_2	Fe_2O_3	Elimination of Fe_2O_3, Pct	CaO, Pct	Elimination of CaO, Pct	MgO, Pct	Elimination of MgO, Pct	Al_2O_3, Pct	TiO_2, Pct	Elimination of SiO_2, Pct
Feed	1.24	97.27	0.030		0.80		0.36		0.09	0.01	
500 lb per hr	0.66	98.58	0.019	22.25	0.41	29.01	0.26	27.50	0.05	0.02	0.20
1000 lb per hr	0.72	98.30	0.036	8.40	0.48	22.80	0.30	23.40	0.09	0.01	0.15
1500 lb per hr	0.86	98.90	0.020	12.40	0.62	15.00	0.42	14.30	0.07	0.02	0.09

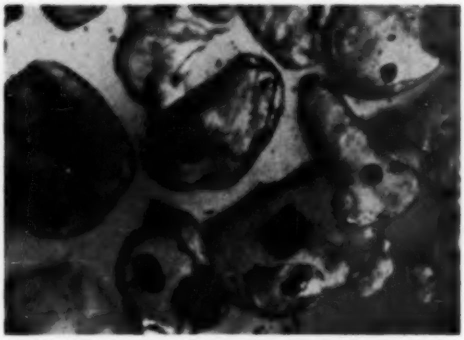


Fig. 4—Photomicrograph of quartz grains showing magnetite inclusions. X57. Area reduced approximately 50 pct for reproduction.

was reduced sufficiently to meet the specifications for chemical sands, and the iron content was reduced to meet the iron specifications for first quality glass sand. However, the MgO plus CaO content was too high to make it acceptable for this purpose.

The Illinois Silica Sand Test: The sand for this test, intended for the glass industry, came from LaSalle County, Ill. Impurity elimination of iron oxide was of prime importance.

Table IV. Chemical Analyses of the Feed and Classified Products Illinois Silica Sand

Test Feed Rate	Loss on Ignition	SiO_2, Pct	Fe_2O_3, Pct	Elimination of Fe_2O_3, Pct	Al_2O_3, Pct	TiO_2, Pct
Feed analysis	0.10	99.77	0.032		0.092	0.10
500 lb per hr	0.11	99.77	0.020	48.20	0.089	0.098
1000 lb per hr	0.12	99.79	0.024	22.20	0.056	0.011
1500 lb per hr	0.11	99.77	0.024	23.90	0.083	0.010

The sand as received was almost pure white and was all −20 mesh in size. The quartz occurred as clear, well-rounded grains, but not smooth. Iron oxide occurred as limonite, 0.04 pct; pyrite, 0.01 pct; and magnetite, 0.04 pct. Ninety-five percent of the limonite occurred as coatings on individual quartz grains. The remainder was in the form of free particles. The ratio of coated quartz particles to clear quartz particles was approximately 1 to 1000. The magnetite was present mainly as inclusions in individual quartz grains. This occurrence was apparently an intergrowth of silica and magnetite, not a deposition of magnetite in cracks in the quartz particles. Noted also were some free magnetite particles. Pyrite occurred mainly as cementing material, but some individual particles were present.

Tests were run at three feed rates, 500 lb per hr, 1000 lb per hr, and 1500 lb per hr. Chemical analyses of the feed and classified products of these tests are shown in Table IV. These analyses showed that the iron oxide content of the sands had been reduced by the sand scrubber. The finished product was observed under the microscope to determine which minerals had been eliminated and to what extent they had been eliminated. It was estimated that about half the coated quartz crystals had been scrubbed clean and the iron oxide eliminated in the classification operation. The pyrite in the form of cementing material had been eliminated, but discrete particles of pyrite and magnetite as well as magnetite in the form of inclusions still remained in the finished product. Fig. 4, a photomicrograph of quartz grains showing magnetite inclusions, readily proves that this form of contaminant could not be removed by a scrubbing operation.

The impurity in the Illinois sand was reduced sufficiently to make the sand acceptable as first quality glass sand.

The power requirement was as follows: 500 lb per hr feed rate, 22 kw-hr per ton; 1000 lb per hr feed rate, 13 kw-hr per ton; 1500 lb per hr feed rate, 10 kw-hr per ton.

Table V. Elimination of Fe_2O_3 Contaminants

Feed Tests	Weight, Pct	Fe_2O_3, Pct	Fe_2O_3 Distribution, Pct
First run		0.45	
Classifier sands	96.5	0.20	38.0
Classifier overflow	3.5	8.90	62.0
Calculated head	100.0	0.51	100.0
Second run		0.20	
Classifier sands	97.3	0.17	77.3
Classifier overflow	2.7	1.9	22.7
Calculated head	100.0	0.22	100.0
Third run		0.17	
Classifier sands	98.5	0.15	93.7
Classifier overflow	1.5	0.67	6.3
Calculated head	100.0	0.16	100.0
200-mesh screening without scrubbing		0.45	
Oversize	93.4	0.35	75.0
Undersize	6.6	1.7	25.0
Calculated head	100.0	0.43	100.0

Ohio Silica Sand: The sand for this test came from near Garrettesville, Ohio, and was an agglomerate containing fine sand, as well as pebbles ranging up to 1 in. in diam. The first three sands described in this paper had a natural grain size below 20 mesh; therefore the procedure worked out for preceding sands had to be altered. The material was first ground in a center peripheral discharge rod mill to −6 mesh. The −6 mesh product was sorted on a screen clothed with 20 mesh stainless steel. The undersize was a mixture of rounded quartz grains

Table VI. Fe_2O_3 Analysis of Feed and Test Products

	As Received	Log Washer	Pebble Mill	Blade Mill	Rod Mill	Paddle Scrubber
Fe_2O_3 in feed, pct	0.247	0.247	0.247	0.247	0.247	0.247
Fe_2O_3 in classified product, pct	0.183	0.176	0.207	0.182	0.137	0.061
Fe_2O_3 in non-magnetic fraction of classified product, pct	0.164	0.138	0.124	0.124	0.107	
Fe_2O_3 in acid leach of non-magnetic fraction, pct	0.123	0.091	0.100	0.107	0.089	0.057
Fe_2O_3 in flotation products, pct						
Classified product as feed		0.107				
Non-magnetic fraction as feed		0.081				
Loss of silica by degradation, pct		3.8	19.0	3.9	3.9	4.7

and angular quartz fragments, brown in color and −20 mesh in size. It was classified in a rake classifier and the sands were fed to the sand scrubber.

The major contaminants were limonite, 0.3 pct, and magnetite, 0.1 pct. More than 50 pct of the quartz grains were coated with limonite, and some occurred as clusters cemented together with limonite. Magnetite was observed as inclusions within the quartz grains and also as individual particles.

It was decided that this material should be run through the scrubber three times, because of its exceptionally high iron content. The material was fed to the scrubber at the rate of 750 lb per hr. The classifier sands from the first run were sampled, then fed back into the scrubber at the same feed rate, and the classifier sands from the re-run were similarly treated. Table V shows metallurgical balances of the test products. A sample of the feed was washed through a 200 mesh screen to give an idea of the cleaning effect of classification alone. The results of this test are also shown in Table V.

Table VII. Metallurgical Balance of Test Products, Nevada Silica Sand

Test	Weight, Pct	Fe_2O_3, Pct	Fe_2O_3 Distribution, Pct	Silica Distribution, Pct
Head		0.247		
Classification				
Classifier sands	96.2	0.193	83.5	96.5
Classifier overflow	3.8	5.355	16.5	3.5
Calculated head*	100.0	0.222	100.0	100.0
Log washer				
Classifier sands	96.0	0.176	68.7	96.2
Classifier overflow	4.0	5.744	31.3	3.8
Calculated head	100.0	0.246	100.0	100.0
Pebble mill				
Classifier sands	81.0	0.207	66.4	81.0
Classifier overflow	19.0	0.447	33.6	19.0
Calculated head	100.0	0.253	100.0	100.0
Blade mill				
Classifier sands	95.9	0.182	76.9	96.1
Classifier overflow	4.1	5.118	23.1	3.9
Calculated head	100.0	0.227	100.0	100.0
Rod mill				
Classifier sands	95.8	0.137	49.5	96.1
Classifier overflow	4.2	5.990	50.5	3.9
Calculated head	100.0	0.265	100.0	100.0
Paddle scrubber				
Classifier sands	95.1	0.061	21.2	95.3
Classifier overflow	4.9	4.383	78.8	4.7
Calculated head	100.0	0.273	100.0	100.0

* Discrepancies in calculated head analysis fall well within limits of experimental error.

After the third treatment, the iron content in this sand was reduced sufficiently to make it acceptable as sixth quality glass sand.

The feed rate to the scrubber was 800 lb per hr for each of the three runs. The power requirement was 15 kw-hr per ton for each run.

Comparison of Several Sand Cleaning Processes

The methods in use today for cleaning sand are mainly washing or classifying operations; however, some include use of gravity shaking tables, blade mills, and blunging. Occasionally ball or pebble mills with light charges of grinding media or rod mills with rubber-coated rods have been used to give a light scuffing action. Further refining has been accomplished by acid leaching, magnetic separation, or froth flotation.

In order to determine the value of the paddle scrubber, the Nevada sand was subjected to various types of cleaning processes. The Nevada sand was selected because a high percentage of its contaminant occurred as a surface coating. The following devices were used as the main beneficiating agent: a log washer, a blade mill, a pebble mill with a light grinding charge, a rod mill with rubber-coated rods, and the paddle scrubber. Products of all of these devices were hydraulically classified with a rake classifier or a constriction plate classifier for slime removal. Some products were refined by acid leach or magnetic separation or by flotation.

Table VIII. Comparative Efficiency of Sand Cleaning Devices

	Sand Scrubber	Log Washer	Pebble Mill	Blade Mill	Rod Mill Rubber Covered Rods	Classifier Only
Feed pct Fe_2O_3	0.247	0.247	0.247	0.247	0.247	0.247
Classified products, pct Fe_2O_3	0.061	0.176	0.207	0.182	0.137	0.193

Iron oxide analyses of the feed and products may be found in Table VI. Both silica distribution and a metallurgical balance for iron oxide in the product are shown in Table VII.

From these tests it was concluded that the paddle scrubber did the most effective job in eliminating surface contamination with minimum degradation; Table VIII shows its superiority over other methods in eliminating iron contamination in Nevada silica sand. Table IX summarizes the performance of the sand scrubber on four sands contaminated by coatings or cementation.

Table IX. Comparative Performance of the Paddle Scrubber on Various Sands

	Contaminant	Contaminant in Feed, Pct	Contaminant in Product, Pct	Elimination of Contaminant, Pct
Nevada sand 1000 lb per hr	Fe_2O_3	0.247	0.061	78.8
Michigan sand 500 lb per hr	CaO	0.80	0.41	29.01
	MgO	0.36	0.26	27.50
	Fe_2O_3	0.030	0.019	32.25
Illinois sand 500 lb per hr	Fe_2O_3	0.032	0.020	48.20
Ohio sand	Fe_2O_3			
800 lb per hr		0.45	0.20	62.0
1st re-run		0.20	0.17	22.7
2nd re-run		0.17	0.15	25.0

The power requirement and capacity of the sand scrubber is governed by the desired product quality. For the tests described here the average power requirement was 14 kw-hr per ton and the capacity ranged as high as 1500 lb per hr. The power requirement for foundry sand reclamation has been found to be as low as 3 kw-hr per ton at a capacity of 5,000 pounds per hr.

These tests prove that the sand scrubber, with accessory equipment, may be used to prepare glass and chemical sands economically from contaminated high silica deposits.

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Variation of Thermo-Electric Properties of Pyrite In Association with Gold Ore

by A. D. Mutch

IT has long been known that there are wide variations in the thermo-electric potential of pyrite. In the last few years a study of this variation and other properties of pyrite was carried out at the University of Toronto by F. G. Smith on specimens of pyrite from many localities.[1-3] As a result of these studies the pyrite geothermometer was developed by Professor Smith.

The calibration of the pyrite geothermometer is based on the assumption that "crystals of any one electronically conducting mineral species deposited at a high temperature have a more positive thermo-electric potential than crystals deposited at a low temperature, and the thermo-electric potential varies continuously between any given limits of the temperature of deposition." The cause of the variation was postulated to be crystal discontinuities, such as lineage boundaries. As yet this postulated relationship has not been established as fact. On theoretical grounds it is indicated that the thermo-electric potential of pyrite or any other partial conductor is dependent upon a number of factors as well as the temperature of formation. For this and other reasons there is ground for doubt as to the validity of the relationship postulated above.

To date a large number of readings have been made with the pyrite geothermometer which have been recorded in terms of the calibration of the instrument in degrees centigrade. So far most of the pyrite tested by this technique has been in association with gold ore. The values for the temperature of formation obtained in these tests appear to be reasonable and, in relation to geological problems, to vary in the right sense. Determinations on some specimens not in association with gold ore have given readings that appear to be much too high, taking into account the probable maximum temperature of the host rock. To sum up, although many determinations made with the pyrite geothermometer appear to give reasonable values for the temperature of formation of the pyrite, there are at the same time other readings which seem anomalous.

On the basis of an examination of available results of work with the pyrite geothermometer, two problems must be considered: 1—the validity of the calibration of the pyrite geothermometer; and 2—the nature of the distribution of the thermo-electric potentials of pyrite in relation to the specific problems that have been examined.

In this paper the academic problem relating to the validity of the calibration of the pyrite geothermometer will not be discussed. Instead, the variations of thermo-electric potential of pyrite in association with gold veins will be discussed with regard to specific examinations.

The first of these two examinations deals with the distribution of thermo-electric potentials of pyrite in association with a gold vein and was carried out in the McIntyre Porcupine Mine, Schumacher, Ont., in the summer of 1949 as part of a study with this technique. The second problem deals with variations of thermo-electric potential of pyrite in gold ore specimens from all over the world. The specimens used in this study were part of the private collection of Frank Ebbutt, geologist for Howe Sound Exploration Co.

Results of Detailed and Reconnaissance Studies

In discussion of the results of determinations made in these two studies the values of thermo-electric potential in millivolts will be used. It will be noted that in some of the diagrams the determinations are recorded in terms of pyrite geothermometer temperatures. Fig. 1 is a calibration chart relating pyrite geothermometer temperatures and thermo-electric potentials.

A full discussion of the technique is given in the original paper by F. G. Smith.[4] Briefly, the determinations are made by placing two stainless steel probes, one hot and the other cold, against the surface of the pyrite crystal being tested. The thermo-electric potential that is developed is read with a null-point potentiometer that is calibrated in what

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is believed by Smith to be the temperature of formation of the pyrite. It can be seen that the technique is simple and that the determinations can be made rapidly.

The operational errors of the instrument may be very large. Because of the small potentials that are measured, changes in the surface conditions between the pyrite and the stainless steel probes cause substantial variations in the readings. The prejudices of the operator also may affect the results in two ways: 1—The instrument is calibrated only at every 100°, and it is necessary to estimate to the nearest 10°. The operator therefore tends to avoid certain values and this in turn affects the averaged results. 2—With two or more groups of readings from one specimen many readings of an intermediate value are obtained and there is a tendency for the operator prematurely to discard some of the readings as anomalous. In the work of the two studies discussed below all readings were recorded and considered.

The data recorded below were obtained with the use of two different pyrite geothermometers. The work around a vein at McIntyre Porcupine Mine was done with one instrument and the work on Frank Ebbutt's worldwide collection of specimens was done with another. As part of the study, not discussed in this paper, carried out by the writer at McIntyre Porcupine Mine the distribution of the pyrite values in relation to the Pearl Lake porphyry was examined. This problem had already been treated in a reconnaissance manner by Smith with the second of the two pyrite geothermometers used in the work discussed below.[5] Comparison of the results of these two studies of the same problem with the two different pyrite geothermometers indicates that because of a personal or instrumental error, the instrument used by the writer on the McIntyre problem read slightly higher than the one used in a study of pyrite in specimens in the Ebbutt collection.

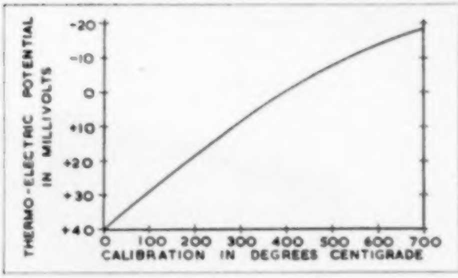


Fig. 1—Calibration curve of the pyrite geothermometer at the University of Toronto.

During the work on both problems, care was taken to reduce the operational errors of the technique to a minimum: 1—all readings were made by the same operator; 2—all readings were taken to the nearest 10°C of the instrument; 3—no results of any phase of the work were plotted until all the determinations were made; and 4—no more than three readings were taken on any one crystal of pyrite.

In the McIntyre work, when enough pyrite was available, up to 20 readings were made on the pyrite in any one specimen. In the work on the Ebbutt collection, a maximum of 10 readings was made on any one specimen.

A detailed description of McIntyre Porcupine Mine is not necessary in understanding this problem. In

Fig. 2—Location of specimens for pyrite geothermometer determinations, No. 25 vein, 3500 level, McIntyre Porcupine mine.

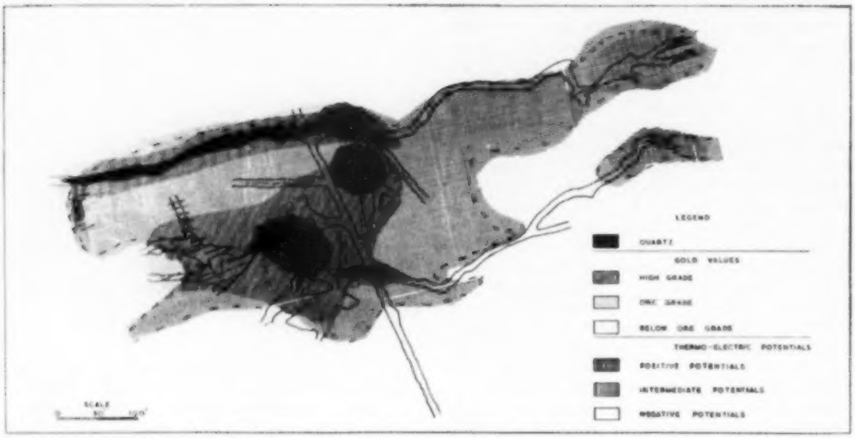


Fig. 3—Variations of the thermo-electric potential of pyrite in association with gold as determined by the pyrite geothermometer, No. 25 vein, 3500 level, McIntyre Porcupine mine.

brief, the mine is a so-called hypothermal gold mine with most of the gold occurring in quartz veins.

For the purpose of examining the detailed variations of thermo-electric properties of pyrite in association with one of these veins, the 3500 level of No. 25 vein was selected. On this level the vein had been well exposed by mining and had been well intersected by diamond drilling. The core from all the drill holes was examined, and where pyrite was found measurements were made with the pyrite geothermometer. A systematic coverage of the drifts was made; specimens were collected and the pyrite in these specimens was tested. As can be seen in Figs. 2 and 3, a good coverage of the vein and the adjoining area was made. Examination of Figs. 2 and 3 indicates a well-defined pattern of distribution of the determinations in relation to the vein areas. Outside the vein area, readings of about −6 mi were obtained, and in the ore shoot, readings of about +22 mi. In the intermediate areas, readings of intermediate and mixed values were obtained.

The Ebbutt collection of gold ore specimens was accumulated by the owner over a period of the last 30 years. In the work discussed below, 462 specimens, of which almost three quarters contained pyrite, were examined and tested with the pyrite geothermometer.

As would be expected, the numerical distribution of the specimens was not truly representative of the distribution of gold mines throughout the world, since 277 of the specimens were from the Canadian Shield, 76 from the province of British Columbia, and the remaining 109 from Nova Scotia, eastern and western United States, Alaska, Mexico, Central and South America, Scandinavia, Europe, Russia, Rhodesia, the Gold Coast, eastern and western Australia, India, the Philippines, and other areas.

Examination of the overall problem indicated that there were enough specimens available from the Canadian Shield to allow an examination of the pattern of distribution of pyrite determinations in relation to the hypothermal type of deposits of this area. It was indicated that there were sufficient specimens from British Columbia so that a similar coverage relative to the Late Mesozoic deposits of this area could be considered. Specimens from the rest of the world were representative of deposits of so many different ages and types that they appeared sufficient to give a confirmatory check if there were any distinct difference between the patterns obtained in the specimens from the Canadian Shield and from British Columbia.

The study of the Ebbutt collection was carried out in two stages. Testing of specimens from the Canadian Shield was done first, followed by work on the specimens from British Columbia. In the testing on the specimens from the Canadian Shield it was found that samples from the mines in the Province of Quebec seldom gave the +22 mi type of reading.

Results of the determinations made in this part of the study were divided into three groups according to the assay of the specimen. Frequency of occurrence curves was plotted with the results of the determinations in these three groups, see Fig. 4. It is noteworthy that in the frequency of occurrence curve for the specimens of low assay, the +22 mi type of reading did not occur.

Results of the second part of the study are shown in Fig. 5 along with the totaled frequency of occurrence curve for the data on the specimens from the Canadian Shield. It will be noted that in these curves there are three peaks, at +22, −6 and −10 mi, that appear to be common to all areas. It is also noteworthy that the −6 mi peak was very high in the curve representing the results of determinations made on specimens from the Canadian Shield and that the +22 mi peak was very high in the curve representing the results of determinations made on the specimens from British Columbia. The relative heights of the peaks on the worldwide curve are intermediate to those on the other two curves.

Critical examination of the results of the determinations combined to make the worldwide curve indicates that the readings obtained on the pyrite

from the many types of deposits varied in the same sense as was expected from differences between the frequency of occurrence curves of the Canadian Shield type of deposit and British Columbia types.

Discussion

Disregarding the postulated relationship between the thermo-electric potential of pyrite and the temperature of formation of that mineral, determinations obtained from the pyrite from all types of gold deposits suggest that all pyrites have similar temperatures of formation.

From an economic standpoint, the pattern of the pyrite determinations obtained in relation to the vein area at McIntyre Porcupine Mine is of the most interest. R. W. Boyle has found that corresponding patterns are indicated at Yellowknife, in the north-west territories of Canada.[6]

Critical examination of the distribution of the pyrite geothermometer determinations in relation to No. 25 vein at 3500 level in McIntyre Porcupine Mine shows that the change in type of determinations can be detected a considerable distance from the vein in the greenstone and porphyry wall rock. When this problem was examined further at McIntyre Porcupine Mine it was indicated that the distance at which the pyrite began to give the +22 mi type of reading of the vein was about 50 ft normal to the vein, about 200 ft along the strike of the vein, and probably a great deal more on the dip. Here, then, may be a technique which will give an indication of the distance from an ore shoot.

There is still much work to be done before the reason for the variation of thermo-electric properties of pyrite is known. The fact that interesting patterns of thermo-electric potentials have been obtained in relation to orebodies suggests that work along these lines should be continued. The problem cannot be fully understood until a thorough study has been made of data on semiconductors that has become available in recent years. It may be that there are other naturally occurring semiconductors, in addition pyrite, which may warrant investigation by the use of a similar technique.

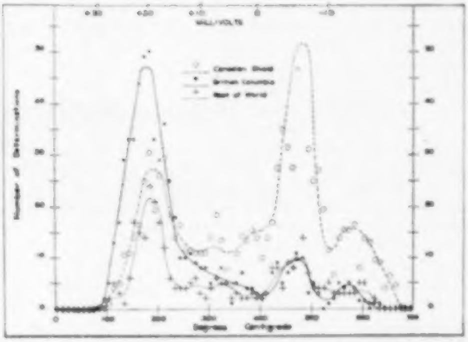
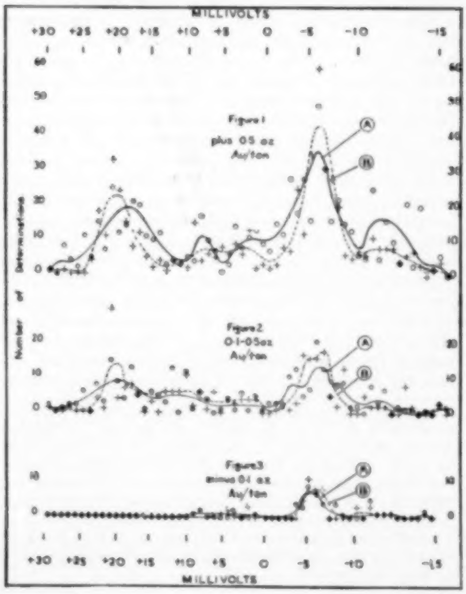


Fig. 5—Frequency of occurrence curves of geothermometer determinations of pyrite in gold ore specimens of the Ebbutt collection.

Fig. 4—Frequency of occurrence curves of pyrite geothermometer determinations in relation to assays of specimens of gold ore from the Canadian Shield. The heavy line and circles indicate determinations recorded, totaled, and plotted irrespective of the sample in which they were found. The dashed line and crosses indicate the totaled and plotted averages of determinations for each specimen, weighted for the number of determinations.



Summary

1—Further research along several different directions is necessary before the qualitative and quantitative aspects of the calibration of the pyrite geothermometer can be reviewed. 2—The frequency of occurrence curves of thermo-electric potentials of pyrite in association with gold ore, representing hypogene gold deposits of all types and all geological ages, show three peaks at +22, −6 and −10 mi. 3—In the frequency of occurrence curve of thermo-electric potentials of pyrite from so-called hypothermal deposits, the highest peak was −6 mi. 4—In the frequency of occurrence of thermo-electric potentials of pyrite from so-called epithermal and so-called mesothermal deposits, the highest peak was +22 mi. 5—In McIntyre Porcupine Mine, Schumacher, Ont., it was found that the +22 mi readings were confined to the orebearing parts of the vein area and that −6 mi readings were confined to the area outside the orebearing parts of the vein. 6—Since the area of the vein containing this modified pyrite is in excess of the ore section, determinations of thermo-electric potential of pyrite may have an economic application in locating and outlining orebodies.

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Drilling and Blasting Methods in Anthracite Open-Pit Mines

by R. L. Ash, R. D. Boddorff, C. T. Butler, and W. W. Kay

DRILLING and blasting in anthracite open-pit mines is a continuous problem to contractors and explosive engineers because of the diverse conditions caused by the nature of the geological formations, the extensive mining of the portions of coal beds near the surface, and the proximity of many strip pits to populated areas.

Pennsylvania anthracite occurs in four separate long and narrow fields totaling only 480 sq miles. The coal measures are rock strata and coal beds that are considerably folded and faulted. The crests of the anticlines are eroded extensively. The beds outcrop on the mountain sides and dip under the valleys. At first only the upper portions of the synclines could be stripped. Now stripping to increasingly greater depths is economically possible, as is indicated by the fact that the proportion of freshly mined anthracite produced by strip mining has increased from 3.7 pct of the total tonnage in 1930 to 29.6 pct in 1950.

Much of the rock overlying the deeper beds now being stripped is so extensively broken that considerable difficulty is experienced in drilling satisfactory blast holes and in using explosives in such manner as to insure a uniformly broken material easily removed by the excavating machinery. Such breaking of rock strata has occurred because the bed now being stripped has been mined extensively in former years by underground methods, and tops of gangways and chambers have subsequently failed.

Draglines are used to uncover coal where the overburden can be moved with little or no rehandling. These machines range in size from those having a 2 cu yd capacity bucket on a 60-ft boom to those handling a 25 cu yd bucket on a 200-ft boom. Draglines are also used to strip to the bottom of the coal basins if the depth and the distance between the crops are not too great. For this type of operation blast holes are drilled full depth to the bed. These holes are commonly 30 to 90 ft deep; however, in exceptional cases, holes may be as shallow as 12 ft or as deep as 130 ft. Drilling is normally done for blasts of 12,000 to 60,000 cu yd of overburden, 30,000 cu yd being considered an average blast if vibration is not the controlling factor.

Where the stripping of wide basins or the exposure of a moderately pitching vein makes the use of draglines impractical, dipper front shovels equipped with 4 to 6½ cu yd buckets load into trucks. Overburden is removed in benches of 25 to 30 ft with blast holes drilled 4 or 5 ft deeper than the planned floor of the bench. For shovels under 5 cu yd bucket capacity the volume blasted varies from 8000 to 12,000 cu yd, whereas a volume of 30,000 to 50,000 cu yd of overburden is frequently blasted at one time for the larger shovels where vibration is not an important factor.

During the past decade the churn drill, generally the Model 42-T Bucyrus-Erie blast hole drill equipped for drilling 9-in. diam holes, has become the most common blast hole drilling machine. Electricity powers half the churn drills in use and is preferred on the large strippings where electric shovels are operated and the working area is concentrated. On these operations the cost of additional electricity for the drills is less than the cost of fuel to operate diesel units because of the existing large demand load of the excavating equipment. Moreover, electric motors start more easily in cold weather and generally are less expensive to maintain. Diesel driven units are employed where a higher degree of mobility is required.

The average drilling speed is 8 ft per hr, although in softer rocks a rate of 15 ft per hr is attained. Where rock is hard and strata is badly broken, drill speeds may be less than 2 ft per hr. Low drilling production results under these circumstances when loose material falling from the upper portion of the drill holes causes drill stems to be jammed.

Rock formations vary so greatly in the region that a 9-in. diam churn drill bit may become dull after drilling only 2 ft or may drill satisfactorily for 56 ft; however, an average of 35 ft is usual in sandstone of medium hardness. Dull bits are hoisted to flat bed trucks by the sand line of the drill and are usually sharpened in the contractor's bit shop adjacent to the job. Care is generally taken to cover the thread end of the bit with a cap. To facilitate handling of bits around the drill, a heavy thread protector having an eye top is becoming more popular than the flat-top rubber or metal cap furnished with new bits.

The 9-in. diam blast holes for a 25 to 30 ft bench are normally on 18x18 ft to 20x20 ft spacings, depending on the character of the overburden, although in broken ground 15x18 ft centers may be used to obtain better breakage and a more even bottom for the bench. The patterns of holes for shots

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to the coal vary from 18x18 ft to 24x24 ft centers because of the generally greater depth of the holes.

In the past two years, 12-in. diam bits with correspondingly heavier drill stems have been substituted for the standard 9-in. tools on some of the Model 42-T blast hole drills. For the most part, the larger holes are drilled where hole depths over 60 ft are required and a dragline with a 22 or 25 cu yd bucket capacity is the excavating unit. By increasing the spacing to 30x36 ft on one operation where a cut ranging from 90 to 130 ft deep across the open face is being taken, 3 drills handling 12-in. tools make available the same volume of shot material as 5 drills with the 9-in. bits.

The hardness and pitch of the rock strata severely limit the application of the horizontal-type auger drills. Nevertheless, in the northern portion of the anthracite region, several contractors have used these drills where the pitch of the vein approaches the horizontal. In one application, a Model 103 McCarthy horizontal drill bored 6-in. diam blast holes to a depth of 36 ft at a speed of 10 ft per hr. The tungsten-carbide cutting teeth were changed before starting each hole because of the abrasiveness of the rock. Holes were spaced 15 ft apart to insure breakage of the 32 ft high, solid sandstone face.

Wagon drills are seldom used for primary drilling, since overburden depths are generally greater than 24 ft, and economical production demands greater volume drilling for large excavating machines.

The Ingersoll-Rand Model QD-8 Quarrymaster percussion drill, equipped with 2 International UD24 diesel engines each driving 412 cfm air compressors to provide air at 150 lb per sq in., was introduced to the region early in 1951. With 35-ft stems and a 6-in. diam bit, these drills have averaged 15 ft per hr in conglomerate and hard sandstone, while twice this rate has been attained in shale and soft sandstone. Production drilling is possible to a depth of 75 ft, but holes have been sunk to 105 ft. Depending upon the material being penetrated, the life of the bit varies from 1500 to over 10,000 ft. The dressing of the spare bits is done at the machine by the operator or his assistant while the drilling continues.

The drilling of blast holes by rotary drilling machinery was started in October 1951, using a Joy Model 58B-H Heavyweight Champion drill, equipped with a 50-hp General Motors diesel engine to drive the rotating table and a 125-hp General Motors diesel engine to operate the 544-cfm air compressor. A 30-ft length of stem can be used without being taken apart; however, standard sections are either 10 or 20 ft long and are screwed together to provide the total length needed. Manufactured by the Hughes Tool Company, the Tri-Cone rock bits used are 7⅞ in. in diam. In hard, broken conglomerate an overall average of 10 ft per hr has been attained with an average bit life of 160 ft. In 20 min, 30 ft have been drilled in shale. When the bit has been used in shale and soft sandstone, a life of 2200 ft has been reached. Experience is limited with respect to drilling deep holes; however, holes are now being drilled to a depth of 65 ft in rock of moderate hardness at a rate of 30 ft per hr.

With the exception of churn drills, all the machines described drill dry holes and deposit fine material near the hole that is suitable for stemming, thus eliminating the costs of supplying water and furnishing stemming material. Bits other than those for churn drills are readily handled by one man without the use of machinery and are discarded or easily resharpened. Because of more continuous contact with the rock to be drilled, the rate of penetration is faster with these machines than with churn drills. To offset these advantages, where air is applied to clean the hole, dust control becomes a serious matter.

Although seemingly high drilling speeds are obtained with some types of equipment, the volume made available for explosives in holes equally deep varies as the square of the diameter of the hole. Thus to produce an equivalent volume per unit time as compared with a 9-in. diam hole drill, a drilling speed 13 times as great is required of a drill making 2½-in. diam holes, 2¼ times for the 6-in., 1½ times for the 7⅞-in., and 0.56 times for the 12-in. diam holes.

Bench shooting is made under comparatively uniform conditions. However, in shooting to the coal, drill patterns of vertical blast holes vary considerably because of the abrupt changes in surface topography and variations in geology caused by the rolls and faults in the tightly folded veins. When drilling is done in broken ground, holes cannot always be drilled to the depth required, since some hit openings and others are abandoned because the upper portion of the holes falls in. Consequently, a planned uniform drill pattern is made irregular by field conditions. Varying depths and burdens, together with strata of vastly different degrees of hardness and brittleness, make it necessary to control carefully the loading of each hole. The burden on the front row often is 24 and 35 ft for 6-in. and 9-in. diam holes respectively, and can be 60 ft for the 12-in. holes at the 130 ft depth.

Because anthracite mining was begun many years ago when a man's home had to be within easy walking distance of his work, towns were built close to the mine openings, directly over the coal basins and near coal outcroppings. Present-day stripping is removing overburden to the very edge of these populated areas, necessitating the utmost caution in blasting to avoid injury to persons or property from flying material and to avoid excessive vibration. Blasts seldom exceed 50,000 lb of explosives for shots to coal, while bench shots average from 8000 to 12,000 lb. However, in some instances it is necessary to place mats over holes and shoot as little as 350 lb of explosives at one time. Several years ago, on the other hand, 660,000 lb of explosives were detonated in a single blast in an area sufficiently removed from property that might be damaged. Heavy shooting is generally required because of the hard and often badly broken overburden. Powder factors range from 0.6 to 1.2 lb of explosives per cu yd of material blasted, but a factor of 0.8 to 0.9 is most common.

Probably one of the most distinguishing characteristics of blasting in the anthracite open-pit mines is the predominant use of relatively insensitive ammonia-type powders in the larger diameter vertical holes. Such powders require special primer charges. Semi-gelatin powder is largely replacing the gelatin, its adoption resulting from the necessity for decreased costs. The semi-gelatins are used almost exclusively for secondary blasting, which is negligible, and for most wagon drill projects.

Explosives are generally delivered directly to blast areas and loaded almost immediately into the blast holes each day. This is made economically possible because of the industry's concentration, the closeness of the powder companies' magazines, and

the large quantities involved with each delivery. Lowering of the large-size powder cartridges is the usual loading practice because the primacord or electric blasting cap wires used for detonating the charges must be protected and because the walls of the blast holes are often jagged. In some circumstances powder is either poured, or dropped, or both, to obtain a denser concentration of the explosives. Often, before loading, holes must be blocked at a desired depth with timber when openings in the coal veins have been encountered. Charges are placed in decks in holes where multiple veins, mine openings, and rock strata of highly differing characteristics are encountered along the depth of the holes. Frequently, to avoid redrilling, holes must be loaded with powder immediately upon being drilled.

For churn drill holes, the surface material immediately available and the breaker slate sometimes used are generally unsatisfactory for stemming, since these materials do not pack properly and the sharp edges of rock so used are likely to damage the primacord or the insulation covering the leg wires if electric caps are used in the hole. Excessive water in stemming is also undesirable, since it prevents proper confinement and causes stemming to be violently dislodged. One large contractor loads silt into a dump truck equipped with a chute so that each hole can be stemmed directly with minimum of labor.

Control of blast by milli-second electric caps or primacord delay connectors is becoming generally accepted and preferred to instantaneous blasting. This type of delay blasting produces less vibration and reduces concussion.

Other advantages over instantaneous blasting include movement of overburden in definite directions, better fragmentation and displacement of rock, and a reduction in overbreak which permits drilling closer to the last row of holes of a previous blast.

The United States Bureau of Mines determined by actual tests[1] that vibrations which did not cause a displacement in excess of 0.05 in. did not cause damage to buildings. The Department of Mines of the Commonwealth of Pennsylvania has established a maximum allowable ground displacement of 0.03 in. at any building to avoid the possibility of damage or undue disturbance. Portable seismographs are frequently used to determine and record the actual ground movement during blasts. The amplitude of vibration is thereby scientifically established and the size of the blasts carefully regulated so as not to create objectionable disturbances.

Acknowledgments

The assistance, co-operation, and guidance of the many anthracite contractors, mining companies, and drill and explosives manufacturers in the preparation of this paper are gratefully acknowledged.

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Drainage Behavior and Water Retention Properties of Fine Coal

by D. W. Gillmore and C. C. Wright

DEWATERING is a major problem in the preparation and utilization of fine-sized coals now being recovered in increasing amounts from colliery effluents, refuse banks, and silt ponds. Of the various methods which have been proposed for dewatering fine coal, gravity drainage is probably the oldest and most widely used, and yet little information on this subject has been published. Consequently a laboratory investigation of gravity drainage phenomena was undertaken not only to provide basic data for predicting the drainage behavior of fine coals in cars, bunkers, basins, and silos, but also to obtain some understanding of the moisture retention properties of fine coal.

Because of practical interest in the problem, a reproducible laboratory technique, which simulated plant conditions as closely as possible, was developed for the evaluation of drainage variables. The apparatus selected consisted of vertically-mounted lucite columns, nominally 4 to 8 ft in length and 2, 3 or 6 in. in diam; the bottom of the column was provided with a rubber stopper through which was inserted a ½-in. diam drainage tube; a wad of glass wool was placed on the upper part of this outlet tube to retain the coal and a plug was temporarily located at the lower end to prevent drainage during charging of the column, see Fig. 1.

The most satisfactory testing procedure is as follows: 1—The fine coal, which has been thoroughly

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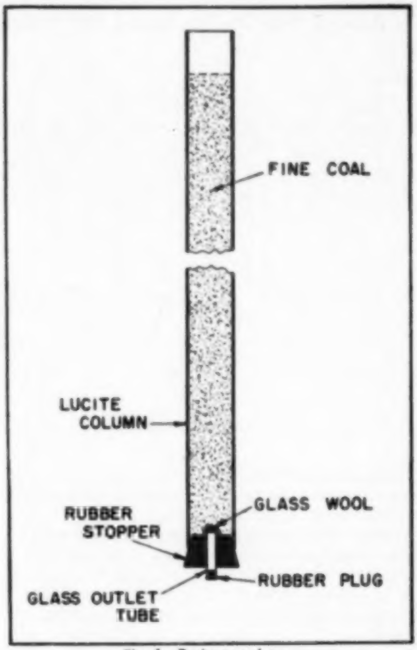

Fig. 1—Drainage column.

wetted by careful mixing and overnight standing, is charged as a slurry of about 50 pct solids in increments of several hundred grams after the glass wool has first been secured with a small amount of moist coal and several inches of water have been added to provide a water pool into which the charge is added. During the charging the column is frequently tapped with a wooden rod or rubber mallet to insure uniform packing and to lessen the occlusion of air. The column is charged to some definite height of coal, e.g., 48 in., and a definite amount of supernatant water is allowed to remain on the coal. 2—Drainage is started by removing the outlet plug and the drainage water is collected in a suitable receiver. When the supernatant water has disappeared, it is advisable to puncture the scum film which forms on top of the draining column and retards drainage. The top of the column should be partially covered during drainage to prevent loss of moisture due to evaporation. 3—The weight of water draining over given times is recorded for the duration of the test so that drainage rates can be computed. 4—At the end of the specified drainage period, usually 24 hr, the column is laid horizontally and, as quickly as possible, small increments of drained coal are cut out successively along the length of the column, starting from the bottom, using a spatula or flat knife. 5—Individual sections are analyzed for total moisture content by determining the percent of weight loss on drying overnight at 100°C. 6—The moisture distribution pattern is obtained by plotting the average height of an increment in the column against the moisture content of that increment.

Equilibrium Moisture Distribution Pattern

A column of drained fine coal normally divides into three zones, a saturation zone at the bottom, a low-moisture zone in the upper section, and a transition zone between these two. In general, the saturation zone is that height of coal in which the moisture content remains at some constant high value substantially equivalent to the saturation value for the particular type of packing. The transition zone is a region of rapidly decreasing moisture content between this saturation zone and a low-moisture zone where the moisture content appears to level out and approach some limiting value. However, for some packings containing a large pore space, such as the anthracite buckwheat sizes, a sharply-defined saturation zone does not exist and the transition zone represents the only section of increased moisture values. Consequently for purpose of discussion throughout this paper the region below the low-moisture zone is frequently referred to as the high-moisture region.

Preliminary drainage tests have shown that for the sizes of coal investigated the moisture distribution pattern is essentially independent of the diameter of the column for diameters of 2, 3 and 6 in. The height of the coal column, for heights up to 8 ft, has no significant effect on the height or moisture content of the saturation or transition zones. Increasing the column height will, of course, decrease the overall moisture content of the column by virtue of lengthening the low-moisture zone.

The rate at which a column of wet coal reaches equilibrium with respect to the formation of the moisture zones has been investigated over a wide range of drainage times. The results for anthracite flotation product* have shown that the pattern of the water-retention equilibrium is established in a few hours and that on further drainage only a small percentage change in the moisture contents or in the heights of the zones is observed. Fig. 2 shows the change in the moisture distribution pattern for drainage times of 3, 12, and 24 hr. The existence of the drainage pattern at shorter times could not be ascertained because of the rapid drainage which occurs during the first 2 hr and the danger of redistributing the water when the column is laid on its side preparatory to sectioning. Additional tests on flotation coal showed that the moisture pattern after 24 days was substantially the same as that obtained

* A typical size analysis for each of the fine coals studied in this program is given in Table I, anthracite, and Table II, bituminous.

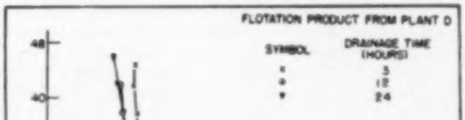

Fig. 2—Effect of drainage time on moisture distribution.

Table I. Typical Size Analysis of Anthracite Fine Coals Used in Drainage Tests, Ro-Tap Method, 200 G Sample, 10 Min

	Weight, Pct											
	Buck No. 1	Buck No. 2, Rice	Buck No. 3, Barley	Buck No. 4	Buck No. 5		Spiral Plant Product	Flotation Products				
					Plant E	Plant D		Plant D	Plant A	Plant B1	Plant B2	Plant E
Screen Size, Round Hole												
+9/16	5.2											
9/16-5/16	74.5	6.8										
5/16-3/16	18.0	50.5	10.4	0.2								
3/16-3/32	2.3*	10.9	67.0	10.5	0.2	0.3						
3/32-3/64		1.3	21.1	74.0	20.2	17.9						
−3/64		0.5	1.5	15.3	73.6	81.8						
Sieve Size, Tyler												
+28				95.1	55.2	47.8	14.7**	16.1	3.3	3.2	2.5	13.3
28x48				4.0	33.5	41.1	56.7**	40.0	43.2	38.9	29.8	37.7
48x100				0.5	8.4	11.1†	21.5	27.5	42.0	37.1	38.3	33.1
100x150				0.1	1.2		2.6	7.7	5.9	8.3	11.4	7.1
150x200				0.1	0.5		1.8	3.8	3.2	6.3	7.4	3.4
−200				0.2	1.2		2.7	4.9	2.4	5.2	10.6	5.4

* −3/16.
† −48 mesh.
** 20 mesh instead of 28 mesh.

in 1 day. It is concluded that the drainage pattern established in 24 hr will not be significantly different even at "infinite" drainage time, and therefore will be considered for purposes of comparison as equilibrium moisture values.

Effect of Particle Size and Size Consist on Drainage Behavior

The drainage characteristics of a column of fine coal appear to be dependent mainly upon the particle size, or mesh size, and the particle size distribution, or size consist, of the material, assuming that the same type of packing is maintained in all the tests. Fig. 3 shows the moisture distribution patterns for a series of closely-sized anthracite fractions, prepared by the multiple screening of dry flotation coal. In general, the height of the high-moisture region and the moisture contents of both the low-moisture and saturation zones increase with decreasing mesh size.

Fig. 4 presents the drainage rates for these selected sizes, plotted as the total moisture content in a 4-ft column of coal at a given time versus the drainage time. The rate and amount of water draining out of a column are shown to increase with increasing particle size. However, in all cases, most of the drainable water is removed during the first few hours and on further drainage a limiting equilibrium moisture content for the entire column is slowly approached. The equilibrium moisture distributions for commercial fine sizes of anthracite are illustrated in Figs. 5 and 6. The drainage rates for some of these sizes are shown in Fig. 7.

In the case of the commercial fine sizes of anthracite the results show that the high-moisture region gradually increases from zero in the case of the coarse product, buckwheat No. 1, to 11 in. for buckwheat No. 5, and to 24 in. or greater for flotation products which contain a substantial percentage of −100 mesh material. The water content of the low-moisture zone appears to increase only slightly with decreasing size either for the more closely sized products, buckwheats No. 1 to No. 4, or for buckwheat No. 5, spiral plant and flotation products, containing a wide distribution of sizes. The high-moisture region increases in moisture content with a decrease in average particle size, but does not show a sharply defined saturation zone until the average size is finer than buckwheat No. 5. The saturation zone for all these smaller sizes generally contains about 35 pct moisture.

Similar drainage tests were performed on various bituminous fines obtained from a central Pennsylvania strip coal. The original sample, prepared to pass a ½-in. screen, was further crushed and screened to give the various size consists shown in Table II.

The results presented in Fig. 8 show the effect of decreasing average mesh size on the moisture distribution pattern in a 4-ft column. In general, the height and moisture content of the saturation zone, as well as the high-moisture region, and the moisture content of the low-moisture zone increase with decreasing mesh size. However, it should be noted

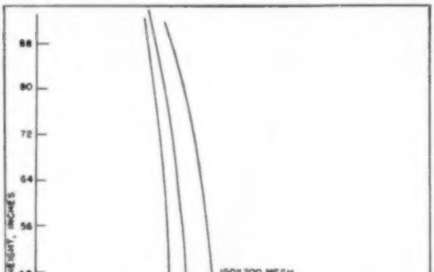

Fig. 3—Moisture distribution for selected sizes of anthracite fine coal, drainage time 24 hr.

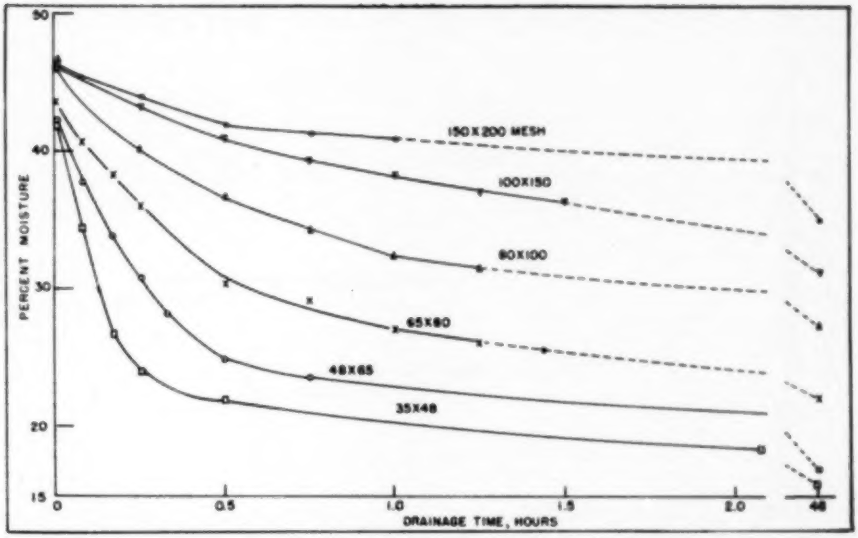

Fig. 4—Change of moisture content with drainage time, selected sizes of anthracite fine coal. Four-foot column.

that with bituminous coal samples, and to a lesser extent with most anthracite sizes, considerable segregation of fines occurred during charging of the coal slurry, causing a sudden increase in the moisture content in the upper few inches of the drained coal. These points have not been plotted in Figs. 8 and 9. It was found that this difficulty could be partly remedied by carefully charging the wet coal into a minimum amount of water in the column. A small water pool is necessary to avoid occlusion of air during charging. Also it was concluded that these drainage tests could be performed more satisfactorily in 8-ft columns to avoid the distracting influence of this "tail" at the top of the column. The slight decrease in moisture values at the lower part of the saturation zone in most tests was due to the admittance of air during drainage and could have been eliminated by the use of a water seal.

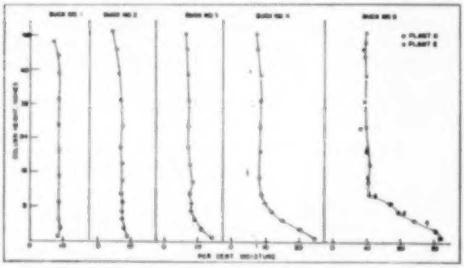

Fig. 5—Moisture distribution for commercial fine sizes of anthracite, drainage time 24 hr.

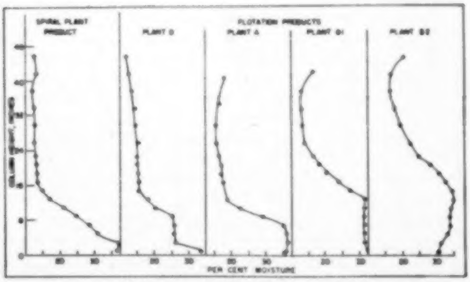

Fig. 6—Moisture distribution for commercial fine sizes of anthracite, drainage time 24 hr. Flotation product, plant A, 48 hr.

Fig. 9 gives the moisture distribution patterns in 8-ft columns for two widely differing size consists, see Table II. The results agree with those described above for 4-ft columns, and with the findings in regard to the effect of size on drainage for anthracite fines. However, bituminous fines exhibit more sharply-defined saturation zones for the coarser sizes; also, these zones increase in moisture content with decreasing average size, possibly because of a wider distribution of sizes in the bituminous fine coals. The results with a wetting agent shown in

Table II. Typical Size Analysis of Bituminous Fines Used in Drainage Tests, Hand-Screened Method, 100 G Sample

Sieve Size, Tyler	FC-96 Original	Weight, Pct				
		FC-96A	FC-96B	FC-96C	FC-96D	FC-96E
+3	22.0					
3x8	34.4	34.1	0.6			
8x14	19.8	23.6	39.6	0.8	29.4	1.7
14x20	6.8	10.9	16.7	1.3	16.6	1.7
20x28	4.8	8.2	11.8	29.4	12.9	15.5
28x48	6.3	11.4	15.5	32.7	19.0	32.7
48x65	1.7	3.2	5.0	11.7	8.9	11.1
65x100	1.6	2.9	3.2	8.6	5.5	11.7
100x150	0.7	1.4	2.0	4.4	3.5	7.4
150x200	0.6	1.0	2.3	3.7	1.7	4.2
−200	1.3	1.2	3.3	7.4	5.5	14.0

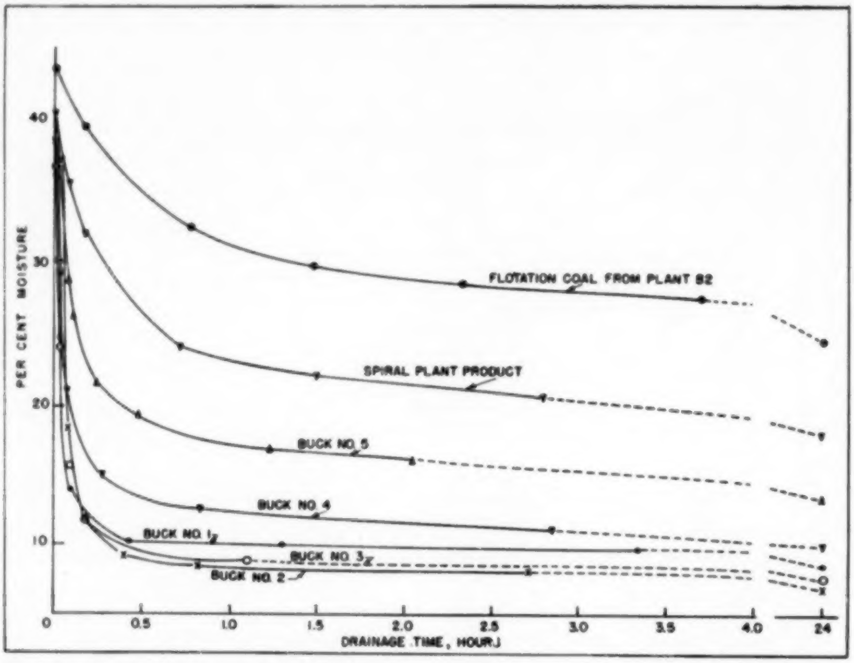

Fig. 7—Change of moisture content with drainage time, commercial fine sizes of anthracite. Four-foot column.

Fig. 9 will be discussed later in the section describing the effect of wetting agents on drainage.

Blending of coarse coal, e.g., 10x16-mesh anthracite equivalent to buckwheat No. 4, with flotation product prior to dewatering results in a decrease in the height of the high-moisture region and a general lowering of the equilibrium moisture values for the entire column, as compared to flotation coal alone, Fig. 10. For example, when a 4-ft column is charged with flotation product, the coal drains to an overall moisture content of 22.4 pct; when 20 pct of the coarse coal is carefully blended with a slurry of fine coal, the overall moisture percentage after drainage drops to 19.0; when 40 pct coarse coal is added, the mixture drains to 16.6 pct overall moisture content.

However, when any appreciable stratification or segregation of the coarse and fine coal occurs through classification or improper mixing, the practice of blending coarse coal with fine coal is of doubtful value in promoting drainage and may actually retard the process. The effect of segregation on the moisture distribution was demonstrated, as shown in Fig. 10, by alternately charging a column with slurries of coarse and fine coal. The drainage results show a pattern of alternating low- and high-moisture layers. Each individual layer represents an independent column of coal in which a drainage pattern is established typical of the size consist of that coal. The overall moisture content of a column charged in this manner was 23.0 pct.

Thus it appears that coarse coal blended with fine coal prior to dewatering effects a lower moisture content in the product, provided that extreme care is taken to avoid segregation of sizes during discharge into the drainage basin, bunker or car.

The removal of undersize, e.g., −200 mesh material, from fine coal also tends to lower the moisture retention. For example, when the percentage of −200 mesh in a flotation coal was reduced from 10.6 to 1.6 by wet screening, the high-moisture region decreased from about 30 to 24 in. and the moisture

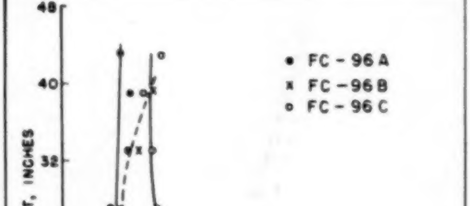

Fig. 8—Moisture distribution for various bituminous fines, drainage time 24 hr. Four-foot column.

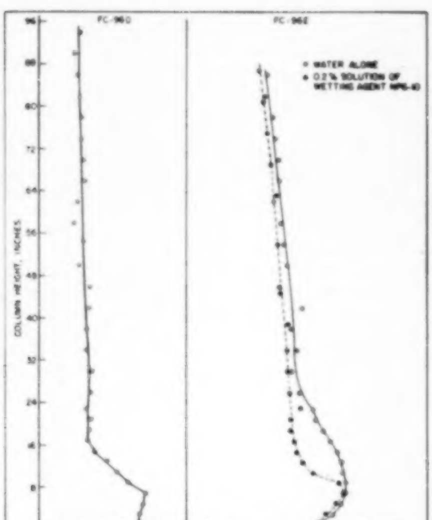

Fig. 9—Moisture distribution for various bituminous fines, drainage time 24 hr. Eight-foot column.

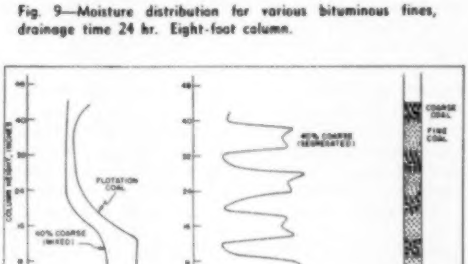

Fig. 10—Effect of blending coarse coal, 10x16 mesh, with the flotation product on moisture distribution, drainage time 24 hr.

content of the low-moisture zone from about 15.0 to 12.5 pct after draining in a 4-ft column for 24 hr; the overall moisture content, however, was lowered only from 24.4 to 22.8 pct.

Experiments Influencing the Design of Bunkers or Basins

Unloading and Recharging Tests: The existence of moisture zones in a column of drained coal immediately suggested the possibility of obtaining a drier product from a drainage bunker, provided that only the coal in the low-moisture zone is removed. Laboratory tests have been made simulating this procedure in an apparatus consisting of a 4-ft, 3-in. ID column, fitted with a flange, to a 2-ft section of the same diameter. The entire column was loaded in the usual manner with flotation product and was allowed to drain for 24 hr. At the end of the drainage period, the 4-ft top section was removed and the coal emptied and analyzed for total moisture content; then the top column was refitted to the undisturbed lower section and was recharged with coal slurry, etc. Twenty-five rechargings were made by this procedure.

Results showed that the moisture content of the replaceable low-moisture section for each of 26 unloadings had essentially the same value, i.e., 18.9 ±1.2 pct. Fig. 11 gives the moisture distribution in the entire column before and after the 25 unloadings and rechargings above the undisturbed high-moisture section. It is of interest to note that the final moisture content for the entire 6-ft column was 23.6 pct, for the low-moisture section, above the flange, 18.5 pct, and for the high-moisture section, below the flange, 32.8 pct.

After the 25 operations, small increments of the high-moisture section were cut out successively along the length of the column and size analyses made on each fraction. The results showed no significant change in size consist in the high-moisture section as compared to either the coal in the low-moisture section or an original sample of the flotation product. Moreover, no plugging or retarding of drainage was observed during the test.

Thus it appears feasible to obtain a drier product from a drainage bunker if the bunker is emptied only above the high-moisture region.

Drainage Leg Experiments: Attempts were made to employ a drainage leg or sump at the bottom of a large column or bunker to learn if such a device would function as an entire high-moisture region, thereby eliminating the high-moisture region above the leg. The results in Fig. 12, for a 4-ft long, 6-in. ID column fitted directly to a 2-ft long, 2-in. ID leg, show a noticeable reduction in the moisture content and in the height of the high-moisture region of the large mass of flotation coal after drainage for either 24 or 48 hr. In a similar experiment, in which the area between the column and the leg was reduced by means of an 8-in. conical section, Fig. 13, no further reduction in the moisture content of the coal in the 6-in. column was observed.

Thus it appears that a leg or sump in the base of a coal column may effectively reduce, but not completely eliminate the high-moisture region in the

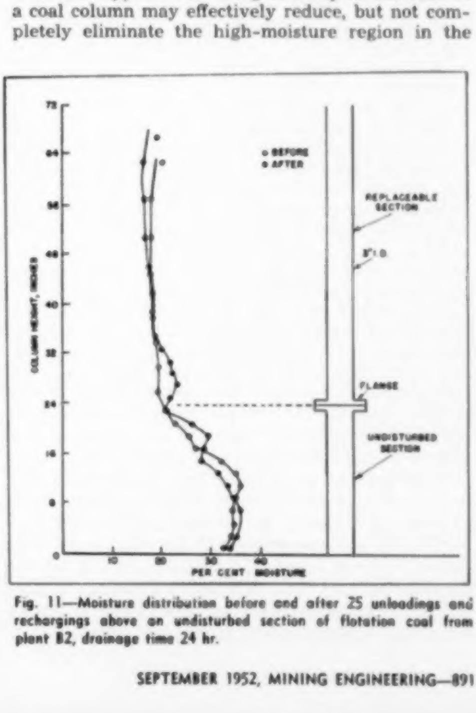

Fig. 11—Moisture distribution before and after 25 unloadings and rechargings above an undisturbed section of flotation coal from plant B2, drainage time 24 hr.

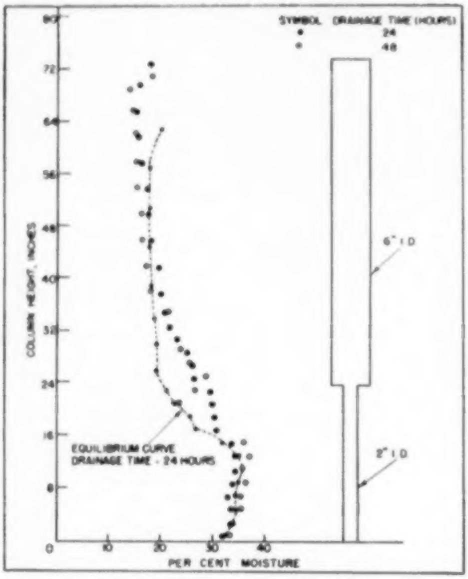

Fig. 12—Effect of drainage leg on moisture distribution of flotation product from plant B2.

main body of the draining coal. It should be noted that the moisture distribution patterns for the columns with a leg were not significantly different from a 6-ft column of a single diameter, see equilibrium curves in Figs. 12 and 13; however, it must be understood that the rate of formation of these patterns and the initial drainage rates were considerably impaired by the nine-fold reduction in cross-sectional area. The effectiveness of any sump or leg arrangement in lowering the moisture percentages also depends on the ability of that device to permit drained water to flow away readily.

Effect of Bedding Material on Drainage: Tests on the effect of bedding material on the drainage of a column of flotation coal indicate that the drainage equilibrium of the fine coal is affected by the nature, particle size, size consist, and thickness of the bed of supporting material. In general, relatively coarse beds do not appear to influence the moisture distribution pattern of the fine coal, but as the size of the supporting material approaches or falls below the size consist of the coal being dewatered, the high-moisture region is greatly reduced and in some cases disappears. Effectiveness of the bedding material in reducing moisture content of the high-moisture region is dependent also on the thickness and moisture distribution of the supporting bed; the optimum thickness is related to the height of the high-moisture region in the supporting bed.

These principles can better be understood by examining Fig. 14 for the effect of three different sizes of bedding material on the drainage of flotation product. The coarse supporting medium, buckwheat No. 4, does not affect the moisture distribution in the flotation coal. Both the fine coal and the coarse bed represent independent columns of coal in which a drainage pattern, typical of the size consist of that coal, is established. However, in the case of the 100x150 mesh or −200 mesh beds, both of which are considerably finer than the flotation coal, the high-moisture region in the flotation product is virtually eliminated, provided that the thickness of the supporting bed is greater than the high-moisture region in that bed. It should be noted that for the buckwheat No. 4 and the 100x150 mesh sizes, bed depths of about 14 and 36 in., respectively, would have given the same moisture distribution in the flotation coal. On the other hand, even a depth of 60 in. of −200 mesh coal was not great enough to allow the formation of a low-moisture zone in the supporting bed. Consequently, the flotation coal above the bed shows a small high-moisture region and slightly higher moisture values in the low-moisture zone. This increase in moisture values was probably also due to the low water permeability of the −200 mesh bed which retarded drainage sufficiently so that in this case equilibrium was not established in the flotation coal within 24 hr.

It is concluded that coarse beds do not affect the moisture distribution pattern of a column of draining coal, but may facilitate dewatering by permitting the drainage water to get away more readily. Also, it appears that beds of fine-sized material do not offer any particular advantage in dewatering and may actually inhibit drainage because of inherent low water permeability or percolation characteristics. The most logical choice of a bedding material, apart from foundation material or drain tile, is a suitable depth of the fine coal being dewatered, the minimum depth being limited by the high-moisture pattern in the coal.

Drainage Test at Settling Basin of Plant B: Flotation product is dewatered at Plant B in either of two basins about 125x125 ft and 15 ft deep, each having a capacity of 5000 to 6000 tons. Coal slurry from the

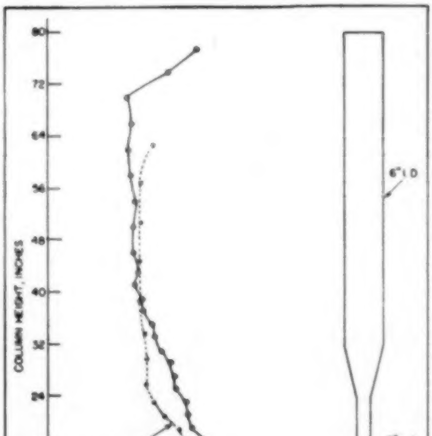

Fig. 13—Effect of drainage leg on moisture distribution of flotation product from plant B2, drainage time 24 hr.

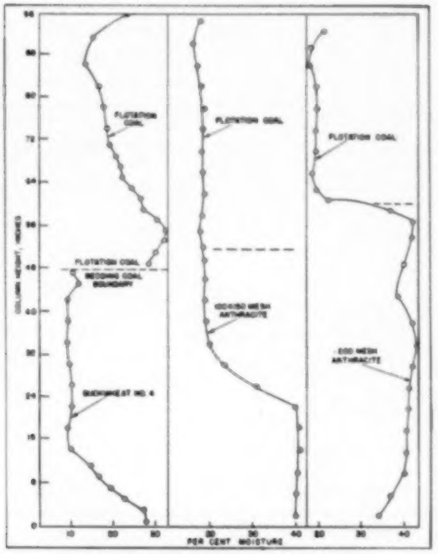

Fig. 14—Effect of bedding material on moisture distribution of flotation coal from plant B2, drainage time 24 hr.

flotation plant at about 25 to 35 pct solids is pumped intermittently for about 30 days, depending upon daily operating schedules, into the center of one of the basins. At the end of this time, the barricade at the portal is removed and the coal is unloaded mechanically while the other basin is being filled.

During the unloading of a basin, the staff members of the operating company removed samples for moisture and size analyses along a vertical face as the unloading machine advanced to three different positions in the basin. The moisture distribution results in Fig. 15 show the presence of moisture zones remarkably similar to those obtained on a laboratory scale. This is good agreement considering the intermittent manner of charging due to starting up and shutting down the flotation plant and to wide variations in solids content and flow rates causing segregation and stratification of material in the commercial basin; this stratification is seen when a vertical face is exposed during unloading.

The screen analyses also show a horizontal classification of the fine coal within the basin; the coarser sizes are found beneath the discharge pipe at the center while some of the fine sizes segregate toward the edge of the basin. It is important to note that the positions of the moisture distribution curves and the overall water percentages for the three sampling points are directly related to the particle size distribution at the respective station. These results give satisfactory evidence that laboratory drainage tests are applicable to plant operating conditions.

Effect of Wetting Agents on Drainage

If it is assumed that the high-moisture region represents the capillary rise of water in a complex bundle of capillaries formed by the packing of the fine coal, then it is an accepted fact that a decrease in the surface tension of the liquid should result in a lowering of the high moisture region. Experiments have been made along this line to demonstrate the existence of a capillarity effect in the lower part of a coal column and to investigate the effect of wetting agents on the drainage behavior of commercial fine coals.

These tests were performed by allowing the fine coal to soak overnight in solutions of the wetting agent and then charging to the column and draining for 24 hr; the drainage results obtained are compared with those from duplicate tests in which no wetting agent was used in the slurry water.

Tergitol NPG-10, experimental-grade, and Tergitol Dispersant NPG, commercial-grade, wetting agents furnished through the courtesy of the Carbide and Carbon Chem. Div., Union Carbide and Carbon Corp., were employed because of the high efficiency of these agents for surface tension reduction at low concentrations. Initial concentrations in solution of 0.005 pct or above will give the minimum surface tension, 35 dynes per cm, compared to distilled water, 75 dynes per cm, as measured by the Du Nouy method.

The results in Fig. 16 show that an initial concentration of 0.1 pct Tergitol NPG-10 in the slurry water decreased the height of the high-moisture region from about 32 to 16 in. and the total moisture content from 26.0 to 20.8 pct in a 4-ft column of flotation coal from plant B2. The surface tension of the drainage water was 40 dynes per cm, indicating that the concentration of wetting agent was only 0.002 pct and that 98 pct of the agent had been adsorbed by the fine coal. In a similar trial with flotation coal from plant E, using Tergitol Dispersant NPG, the high-moisture region was lowered from about 22 to 10 in. and the overall moisture content of the 4-ft column from 22.5 to 18.7 pct; the drainage water again showed less than 0.002 pct wetting agent in solution.

The effect of wetting agents on the drainage behavior of bituminous fines has been investigated briefly as shown previously in Fig. 9. Sample FC-96E was allowed to soak overnight in a 0.2 pct solu-

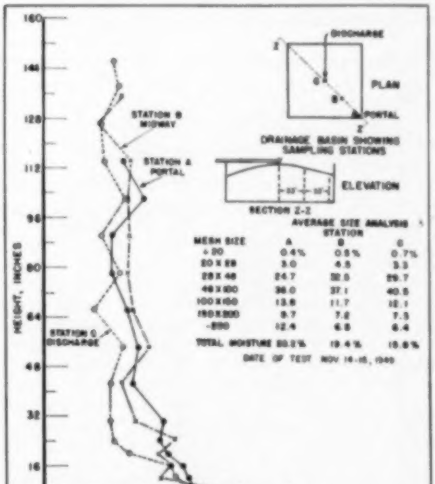

Fig. 15—Drainage tests at flotation settling basin of plant B.

tion of Tergitol NPG-10 and the moisture distribution pattern was determined after drainage for 24 hr. The initial concentration of wetting agent was sufficient to give the minimum surface tension, 35 dynes per cm, before and during drainage. The adsorption of the wetting agent by the bituminous fines was not determined.

Results for spiral plant product were equally encouraging in so far as moisture reduction was concerned, showing a decrease in the total moisture content of the coal from 17.9 pct with distilled water alone to 16.6 pct with 0.01 pct solution of the wetting agent, and to 13.8 pct with 0.1 pct solution, see Fig. 17. However, in the case of the test with 0.01 pct initial solution, considerable adsorption of the wetting agent by the coal was noted, as the surface tension increased from 35 to 66 dynes per cm; this change represented a decrease in concentration of the wetting agent from 0.01 pct to less than 0.0002 pct. It was observed that adsorption of the wetting agent by the fine coal occurred principally during the first few minutes of contact between the solution and the fine coal, and was not necessarily aggravated by allowing the mixture of fine coal and solution to stand overnight before charging to the column. This adsorption of wetting agent imposes a serious limitation on the use of such agents for lowering surface tension in coal slurries and thus for reducing the moisture content of a column of draining coal. Undoubtedly the degree of adsorption of the wetting agent is related to the nature of the agent and to the surface area of the fine coal. The adsorption may, therefore, be expected to vary markedly with size consist and probably also with the wetting agent used.

The drainage rate data for flotation and spiral plant products showed that the addition of a wetting agent does not materially alter the drainage rates during the early stage of a test, where most of the drainable water is removed and where the moisture distribution pattern is being established. However, wetting agents lower the moisture retention of the

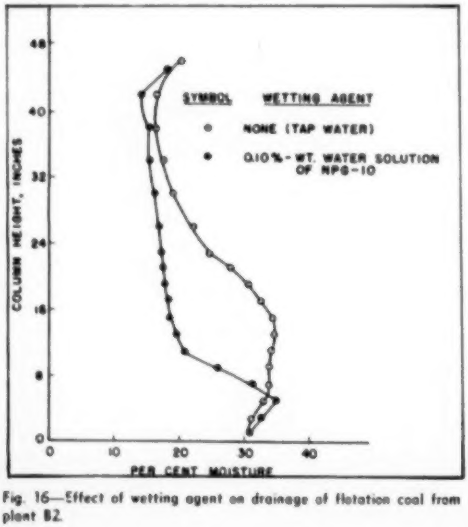

Fig. 16—Effect of wetting agent on drainage of flotation coal from plant B2.

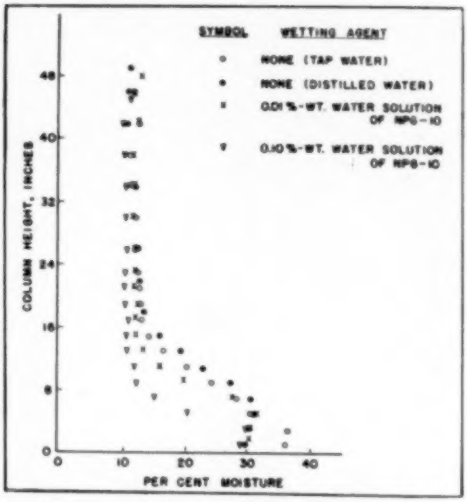

Fig. 17—Effect of wetting agent on drainage of spiral plant product, drainage time 24 hr.

fine coal and do increase the rate at which the lower equilibrium moisture values are approached.

For the particular size consist and wetting agent employed, an initial concentration of at least 0.1 pct wetting agent in solution is necessary, i.e., about 2 lb of agent per ton of dry coal, in order to offset the adsorption by the fine coal and still produce a low enough surface tension for lowering the moisture values of flotation coal. At the prevailing cost of about 35 cents per lb, the use of this wetting agent does not appear to be economically feasible. Dousing or spraying the top of a column of drained or draining flotation coal with solutions of wetting agent did not significantly alter the quantity of wetting agent required to effect a definite moisture reduction.

However, for coarse materials, where the surface area is only a fraction of that exposed by flotation coals, e.g., buckwheat No. 4 has about one-fifth the surface area of flotation coal per unit weight, the adsorption of the agent would be greatly reduced and the application of wetting agents might be more desirable. Also, the recycling of the drainage water would appear more feasible for the coarser sizes with the resulting saving in wetting agent required to maintain a low surface tension in the coal slurry. Nevertheless, it should be noted that the principal function of the wetting agent is to lower the height of the high-moisture region in the draining coal, and that the effectiveness of the wetting agent in lowering overall moisture contents would be greatly diminished for sizes which already show a low high-moisture region with water alone.

Acknowledgments

The authors are deeply indebted to J. D. Clendenin and H. J. Donald for many helpful suggestions and comments in the pursuance of the work, to G. W. Jones, M. Moul, A. J. Peterson, and J. H. Schalch for assistance in the experimental work, to E. L. Maloy and M. Ott for preparing the drawings, and to the Anthracite Research Advisory Committee for permission to publish the drainage results on anthracite fine coals.

Comparative Effectiveness of Coal Cleaning Equipment

by Orville R. Lyons

This paper presents a method whereby the amount of misplaced material and the difficulty of the separation can be used to compare coal cleaning equipment of all types, from effectiveness and capacity standpoints. The correlations presented do not include all types of equipment currently available, but the method can be used to evaluate any make or type of coal cleaning equipment, both old and new.

THE relative performance of coal washing equipment, or the effectiveness with which any type or make of equipment removes impurities from coal, has been most difficult to evaluate in the past. The most widely used yardstick is the Frazer and Yancey efficiency formula developed in 1922,[1] but Yancey in a later article states that "washers treating coals of different density composition or operating at different densities of separation cannot be compared directly on the basis of this criterion."[2] Prior to and since 1922, a variety of other methods has been used for comparison purposes, including the distribution curve, the error area, and the "ecart probable" or probable error. Yancey and Geer in discussing these methods conclude, "Performance can be evaluated in a number of different ways, with the choice of the proper method to use being dictated by the objectives of the investigation and the data available."[2]

It is true that performance can be evaluated in a variety of ways, but if the equipment is to be evaluated on an effectiveness basis, there should be only one universal comparison method. Varying methods have been used because one universal comparison method has not been found or developed.

In the article previously quoted, Yancey and Geer state in clear terms the primary concept for a universal comparison method: "One of the simplest, and certainly one of the most obvious evaluations of washery performance is the quantity of sink material in the washed coal and the float material in the refuse. If the washery products are tested at the density at which the washing unit is operated, the sink in the washed coal and the float in the refuse represent material that has been misplaced."

The quantity of misplaced material was used as a criterion of washery performance by Lincoln in 1913,[3] by the United States Bureau of Mines in 1938,[4] by Hancock in 1947,[5] and by the national French research agency Cerchar in recent years.[6] In 1950 Anderson[7] proposed the use of this criterion as an efficiency value to replace the Frazer and Yancey formula. However, none of the above-mentioned investigators used the misplaced material concept in a manner that would provide universal coal-cleaning equipment comparisons.

The Correlation Theory

The ideal coal cleaning process would treat all sizes and would make a perfect separation at any given specific gravity. All material lower in density than the desired value would report in the coal product and all material higher in density would report in the refuse product. Unfortunately, no known cleaning process achieves this goal and there seems little likelihood that any process yet to be invented will do more than approach it.

When coal is treated in volume under operating conditions, it is impossible to avoid mechanical entrapment, fluctuations in throughput and effective gravity of separation, and the creation of turbulent currents, even when a true heavy-liquid bath is used and the feed is closely sized and contains little intermediate gravity material. This being so, it is possible to appreciate the difficulties inherent in trying to obtain a perfect separation when treating a wide range of sizes and a feed containing high percentages of intermediate material, using turbulent currents to help create the effective separation gravity, under operating conditions which normally tend to be on the overload side.

When coal is separated from refuse in any coal cleaning equipment, some refuse always reports to the coal and some coal to the refuse; the writer therefore assumed that there should be a relationship between the total amount of misplaced material produced by any given piece of equipment and the difficulty of separation as represented by the percentage of near gravity material in the feed. With small amounts of near gravity or ± 0.1 material in the feed there should be less misplacement of material than would occur with large amounts of near

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Table I. Summary of Operating Data for a Variety of Coal Cleaning Equipment

Plant No.	Cleaning Unit: Type	Cleaning Unit: Make	Coal Treated: Seam	Coal Treated: Size, In.	Washed Coal: Wt, Pct	Washed Coal: Sink, Pct	Refuse: Wt, Pct	Refuse: Float, Pct	Gravity of Separation	±0.1 Distribution Pct at Gravity of Separation*	Total Misplaced Material, in Washed Coal and Refuse, in Pct of Raw Coal
1	Sand media	Chance	Illinois No. 6	6x1½	58.00	2.55	42.00	11.71	1.400	57.5	6.40
2	Sand media	Chance	Great Britain	5x1/16	78.50	1.19	21.50	2.60	1.425	24.0	1.49
3	Sand media	Chance	Elkhorn	4x⅛	80.05	1.30	19.95	4.81	1.440	22.7	2.00
4	Sand media	Chance	Illinois No. 6	1½x5/16	86.00	0.60	14.00	0.90	1.450	14.7	0.65
5	Sand media	Chance	Lower Freeport	3½x⅜	88.00	0.50	12.00	0.50	1.450	15.5	0.50
6	Sand media	Chance	Upper and Lower Freeport	¾x⅛	65.00	0.50	35.00	2.25	1.450	25.0	1.12
7	Sand media	Chance	Thick Freeport	3½x¼	76.00	1.50	24.00	1.50	1.440	27.2	1.50
8	Bone media jig	Jeffrey	Upper Freeport	2x0	80.00	7.00	20.00	2.50	1.400	55.0	6.10
9	Bone media jig	Jeffrey	Upper Freeport	2x0	80.00	1.00	20.00	8.00	1.430	31.0	2.40
10	Bone media jig	Jeffrey	Pocahontas 3 & 5	7x⅝	80.00	3.50	20.00	6.75	1.475	33.0	4.15
11	Magnetite media	Link belt	Anthracite	4⅜x3/16	68.10	1.60	31.90	3.04	1.732	33.0	2.06
12	Fine iron media	Tromp	France	2x½	59.56	0.07	40.44	0.25	1.750	5.0	0.15
13	Magnetite media	Tromp	The Netherlands	3x¾	77.35	0.11	22.65	0.05	1.600	4.0	0.10
14	Magnetite media	Tromp	Great Britain		80.15	0.30	19.85	8.30	1.350	5.3	1.89
15	Fine iron media	Tromp	Great Britain		85.60	0.16	14.40	0.00	1.55	6.22	0.14
16	Fine iron media	Tromp	Great Britain		76.57	0.99	23.43	7.46	1.40	30.20	2.51
17	Fine iron media	Tromp	Great Britain		81.21	0.26	18.79	0.00	1.60	6.40	0.21
18	Pulsator jig	Elmore	Wilkeson	⅜x3/16	78.00	15.56	22.00	10.79	1.50	29.60	14.50
19	Pulsator jig	Elmore	Wilkeson	⅜x3/16	61.50	6.95	38.50	31.10	1.50	29.60	16.23
20	Classifier	R & S hydroseparator	Penna. Pittsburgh	5x1	68.75	1.53	31.25	17.95	1.50	9.40	6.66
21	Classifier	R & S hydroseparator	Penna. Pittsburgh	1x5/16	87.85	1.57	12.15	13.70	1.70	4.65	3.04
22	Classifier	Menzies hydroseparator	Pocahontas No. 4	2½x1	85.00	7.20	15.00	23.70	1.44	14.30	9.68
23	Classifier	Menzies hydroseparator	Penna. Pittsburgh	4x¾	87.30	0.75	12.70	10.75	1.60	2.60	2.02
24	Classifier	Menzies cone	Illinois No. 6	3x2	86.20	1.35	13.80	11.50	1.55	9.00	2.75
25	Classifier	Menzies cone	Illinois No. 6	1½x¾	89.28	3.10	10.72	12.81	1.50	9.85	4.15
26	Classifier	Menzies cone	Illinois No. 6	2x1½	81.90	3.50	18.10	13.60	1.50	13.60	5.32
27	Classifier	Menzies cone	Eagle	2¾x⅜	85.00	3.60	15.00	6.20	1.42	17.50	3.99
28	Classifier	Menzies cone	Pocahontas No. 3	2½x⅝	81.00	3.45	19.00	21.40	1.48	26.00	6.86
29	Classifier	Menzies cone	Eagle	5x1¼	85.00	0.50	15.00	6.20	1.55	5.50	1.36
30	Classifier	Menzies cone	Illinois No. 6	¾x⅜	91.30	1.20	8.70	15.80	1.70	4.55	2.47
31	Classifier	R & S hydrotator	Anthracite	13/16x28†	88.10	3.20	11.90	4.70	1.70	6.50	3.38
32	Classifier	R & S hydrotator	Miller	1½x⅛	71.00	1.14	29.00	1.49	1.60	4.00	1.24
33	Classifier	R & S hydrotator	Penna. Pittsburgh	⅛x48†	95.50	1.06	4.50	15.36	1.55	2.96	1.47
34	Classifier	R & S hydrotator	Penna. Pittsburgh	9/16x28	72.50	2.50	27.50	3.50	1.50	5.80	2.77
35	Wet table	Deister Conc. Co.	Island Creek	⅜x0	88.67	3.40	11.33	3.70	1.60	10.00	3.44
36	Wet table	Deister Conc. Co.	Penna. Pittsburgh	⅜x100†	91.60	11.84	8.40	5.75	1.35	45.50	11.32
37	Wet table	Deister Conc. Co.	Penna. Pittsburgh	⅜x100†	93.00	0.90	7.00	5.72	1.60	2.40	1.24
38	Wet table	Deister Conc. Co.	Ohio Pittsburgh No. 8	3/16x100†	77.90	3.50	22.10	4.70	1.50	14.60	3.77
39	Wet table	Deister Conc. Co.	Ohio Pittsburgh No. 8	3/16x100†	77.90	1.80	22.10	8.50	1.60	6.60	3.28
40	Wet table	Deister Conc. Co.	Pratt	¼x0	86.80	13.70	13.20	4.40	1.40	24.00	12.50
41	Wet table	Deister Conc. Co.	Pratt	¼x0	86.80	1.05	13.20	7.00	1.60	2.70	1.83
42	Wet table	Deister Conc. Co.	Penna. Pittsburgh	½x0	81.20	4.45	18.80	9.90	1.50	16.90	5.48
43	Wet table	Deister Conc. Co.	Penna. Pittsburgh	½x0	81.20	2.80	18.80	11.10	1.60	4.50	4.36
44	Wet table	Deister Conc. Co.	Upper & Lower Freeport	¼x0	91.20	1.40	8.80	12.90	1.60	6.10	2.41
45	Air table	Stump	Penna. Pittsburgh	⅜x0	67.80	11.02	11.11	32.20	1.60	4.00	15.41
46	Air table	Stump	Roslyn	⅜x0	91.00	9.20	9.00	68.70	1.60	3.14	14.54
47	Air table	Stump	Roslyn	⅜x0	91.00	15.50	9.00	59.10	1.40	40.60	19.42
48	Air table	Stump	Penna. Pittsburgh	⅜x0	67.80	12.57	32.20	22.62	1.50	9.00	15.80
49	Air table	Stump	Penna. Pittsburgh	⅜x0	67.80	9.40	32.20	26.98	1.70	4.40	15.05
50	Air table	Stump	Lower Elkhorn	¼x40†	82.70‡	15.90	17.30	46.20	1.44	12.50	21.12
51	Air table	American		½x¼	89.40	2.97	10.60	62.80	1.70	2.50	9.43
52	Air table	American	Roslyn	¼x0	93.53	16.00	6.47	37.90	1.40	32.80	17.45
53	Air table	American	Roslyn	¼x0	93.53	9.50	6.47	48.00	1.60	3.83	12.01
54	Barytes bentonite media	Dutch cyclone	Penna. Pittsburgh	14x35†	95.70	0.20	4.30	6.25	1.63	1.54	0.46
55	Magnetite media	Dutch cyclone	Penna. Pittsburgh	¼x0	88.92	1.12	11.08	5.47	1.55	2.63	1.60
56	Potash media	Dutch cyclone	European	5/16x16†	66.70	4.10	33.30	4.38	1.80	14.43	4.19
57	Loess media	Dutch cyclone	European	19/32x0	86.25	1.99	13.75	2.70	1.45	19.20	2.09
58	Roasted pyrite media	Dutch cyclone	European	5/16x32†	77.20	9.99	22.80	8.01	1.70	47.50	9.54
59	Loess media	Dutch cyclone	European	7/64x32†	88.30	2.28	11.70	2.59	1.45	32.60	2.31
60	Loess media	Dutch cyclone	European	19/32x0	86.25	5.78	13.75	1.34	1.40	81.00	5.17
61	Roasted pyrite media	Dutch cyclone	European	5/16x32†	77.20	1.19	22.80	21.76	1.80	14.00	5.88
62	Loess media	Dutch cyclone	European	7/64x32†	88.30	0.82	11.70	6.85	1.50	8.70	1.52
63	Barytes media	Dutch cyclone	Landau No. 3	¼x20†	41.30	20.30	58.70	10.90	1.35	72.60	14.78
64	Barytes media	Dutch cyclone	Landau No. 3	¼x20†	44.50	18.90	55.50	9.50	1.35	72.60	13.68
65	Barytes media	Dutch cyclone	Landau No. 3	¼x20†	60.00	13.40	40.00	17.30	1.40	70.20	14.96
66	Barytes media	Dutch cyclone	Coronation	¼x20†	41.00	14.40	59.00	33.00	1.40	84.00	25.37
67	Barytes media	Dutch cyclone	Spring-bok	¼x20†	26.70	19.30	73.30	23.80	1.35	76.00	22.60
68	Barytes media	Dutch cyclone	Spring-bok	¼x20†	45.40	32.50	54.60	8.60	1.35	76.00	19.45
69	Barytes media	Dutch cyclone	Phoenix	¼x20†	30.30	15.90	69.70	15.80	1.35	71.90	15.83
70	Barytes media	Dutch cyclone	Phoenix	¼x20†	63.00	11.70	37.00	7.40	1.40	74.40	10.11
71	Barytes media	Dutch cyclone	Phoenix	¼x20†	50.50	29.70	49.50	6.70	1.35	75.70	18.31
72	Spiral	Humphreys	Pocahontas No. 6	8†x0	96.30	3.10	3.70	52.60	1.60	1.78	4.93
73	Spiral	Humphreys	Clements	8†x0	96.50	3.20	3.50	42.10	1.60	0.74	4.56
74	Spiral	Humphreys	Roslyn	8†x0	85.3	7.70	14.70	27.60	1.60	6.3	10.63
75	Spiral	Humphreys	Kentucky No. 9	8†x0	94.3	14.90	5.70	21.20	1.60	5.8	15.26
76	Spiral	Humphreys	Raton & Trinidad	8x100†	77.7	60.30	22.3	3.10	1.38	67.5	47.49
77	Spiral	Humphreys	Raton & Trinidad	8x100†	77.7	14.20	22.3	14.70	1.60	8.7	14.28
78	Wet table	Deister Mach. Co.	Raton & Trinidad	½x0	86.95	15.60	13.05	4.70	1.38	27.5	14.21
79	Wet table	Deister Mach. Co.	Raton & Trinidad	½x0	86.95	4.10	13.05	13.20	1.60	6.53	5.32
80	Wet table	Deister Mach. Co.	Raton & Trinidad	½x0	93.08	9.20	6.92	4.50	1.38	21.90	8.87
81	Wet table	Deister Mach. Co.	Raton & Trinidad	½x0	93.08	1.90	6.92	12.50	1.60	3.60	2.64
82	Bone media jig	Jeffrey	Raton	6x⅜	82.00	0.45	18.00	5.20	1.60	5.66	1.31
83	Bone media jig	Jeffrey	Penna. Pittsburgh	4x¼	90.33	0.83	9.67	2.74	1.55	2.40	1.01
84	Pulsator jig	Vissac	Roslyn	3x1¼	78.00	4.50	22.00	17.50	1.45	25.00	7.36
85	Pulsator jig	Vissac	Roslyn	1¼x¼	92.56	1.50	7.44	18.00	1.69	3.00	2.73
86	Pulsator jig	Jeffrey	Routt	3x1¼	95.63	1.40	4.37	10.90	1.70	1.46	1.82
87	Pulsator jig	Jeffrey	Routt	3x1¼	95.63	1.80	4.37	8.10	1.60	2.29	2.07
88	Mill scale media	Ridley-Scholes	Great Britain	3x¼	86.00	1.79	14.00	0.40	1.45	10.00	1.60
89	Mill scale media	Ridley-Scholes	Great Britain	3x¼	89.00	0.59	11.00	0.50	1.60	4.46	0.56
90	Classifier	Menzies cone	Illinois No. 6	1½x¾	95.37	2.69	4.63	17.67	1.60	8.50	3.48

Table I. Summary of Operating Data for a Variety of Coal Cleaning Equipment (Continued)

Plant No.	Cleaning Unit: Type	Cleaning Unit: Make	Coal Treated: Seam	Coal Treated: Size, In.	Washed Coal: Wt, Pct	Washed Coal: Sink, Pct	Refuse: Wt, Pct	Refuse: Float, Pct	Gravity of Separation	±0.1 Distribution Pct at Gravity of Separation*	Total Misplaced Material, in Washed Coal and Refuse, in Pct of Raw Coal
91	Classifier	Menzies cone	Illinois No. 6	2x1½	88.80	5.31	11.20	4.53	1.40	21.00	5.23
92	Classifier	Menzies cone	Illinois No. 6	2x1½	88.80	0.42	11.20	16.80	1.50	13.40	2.22
93	Classifier	Menzies cone	Illinois No. 6	¾x⅜	91.67	1.70	8.33	10.58	1.60	5.47	2.44
94	Pulsator jig	Elmore	Wilkeson	3/16x3/32	69.30	11.43	30.70	24.10	1.50	25.00	15.34
95	Pulsator jig	Elmore	Wilkeson	1½x⅜	79.90	3.85	20.10	17.69	1.70	12.50	6.64
96	Pulsator jig	Elmore	Wilkeson	1½x⅜	56.10	10.86	43.90	20.35	1.50	31.40	17.69
97	Magnetite media	Link-Belt	Penna. Pittsburgh	3x⅜	96.10	1.20	3.90	1.90	1.60	9.70	1.22
98	Magnetite media	Link-Belt	Penna. Pittsburgh	4x⅜	52.80	0.40	47.20	1.10	1.60	9.60	0.73
99	Magnetite media	Link-Belt	Penna. Pittsburgh	4½x⅛	68.30	0.70	31.70	1.70	1.55	4.75	1.02
100	Magnetite media	Link-Belt	Penna. Pittsburgh	4x5/16	29.50	9.50	70.50	6.80	1.35	61.00	7.40
101	Magnetite media	Tromp	The Netherlands	3½x⅜	48.3	0.40	51.7	0.40	1.535	20.00	0.40
102	Magnetite media	Tromp	The Netherlands	3½x⅜	57.0	0.50	43.0	0.50	1.835	8.00	0.50
103	Magnetite media	Tromp	The Netherlands	¾x¼	58.22	6.15	41.78	0.007	1.40	64.00	3.67
104	Magnetite media	Tromp	The Netherlands	¾x¼	58.22	0.14	41.78	1.31	1.45	55.00	1.40
105	Magnetite media	Tromp	France	2x5/16	46.78	1.10	53.22	1.01	1.75	46.80	1.08

* Corrected to allow for sink 2.0 sp gr material in raw coal.
† Mesh.
‡ 4 pct of raw feed was trapped by deduster and added to final washed coal; therefore, washed coal and refuse products shown here represent only 96 pct of raw feed.

gravity material. In both cases, naturally, the machine would need to be adjusted to provide a maximum separation of coal from refuse.

Such a relationship, if it could be developed, would include the three criteria listed by Yancey and Geer[2] as being essential to a comparison evaluation. The percentage of ±0.1 near gravity material in the raw coal at the gravity of desired separation would adequately represent the character of the raw coal and the density of the separation. The total amount of misplaced material, expressed as a percentage of the raw feed, would represent the effectiveness of separation between coal and impurity.

The Correlation Data

Table I contains a summary of data obtained for a total of 105 large scale tests on a great variety of coal cleaning equipment. These data were obtained from the literature,[1-20] from operators and equipment manufacturers,[21-27] and from tests conducted at preparation plants of Republic Steel Corp. The major portion of the data represents tests conducted at preparation plants under normal operating conditions. The remainder of the data is for tests conducted on a pilot-plant scale. In so far as could be determined, all the tests represent careful sampling and accurate analyses. All the raw coal float-and-sink data were replotted and the ±0.1 near gravity values adjusted to compensate for the amount of sink 2.0 specific gravity material in the feed, thus placing all of the data on a common basis.

Table II contains data showing the range of separating gravities at which separations were actually made by the various types and makes of coal cleaning equipment. No claim is made that the range of gravities shown in Table II is complete, as some of the equipment listed is being used to make separations at gravities lower and higher than shown.

Coarse Coal Cleaning Units: Figs. 1 and 2 present in graphical form correlations for coarse coal cleaning units or units used to treat coals having top sizes in excess of ½ in. with no limitation on the bottom size. The coarse coal units for which data were available include: 1—the Tromp process, 2—the Chance cone, 3—the Link-Belt drum, 4—the Ridley-Scholes bath, 5—the Jeffrey-Baum jig operated with a bone recirculation, 6—the Jeffrey diaphragm jig, 7—the Vissac pulsator jig, 8—the Menzies cone, 9—the Roberts & Schaefer-Menzies Hydroseparator, and 10—the Elmore pulsator jig. These data were plotted on two separate graphs to avoid overlapping and to show to best advantage the correlations obtained.

Of all the equipment for which test data were obtained, the Tromp dense-media bath is the most effective separator of coarse refuse from coal. The Chance cone is just as effective as the Tromp bath when treating feeds containing less than 7 pct ±0.1 near gravity material but is less and less effective as the difficulty of the separation increases.

The Link-Belt dense-media drum is slightly less effective than the Chance cone and Tromp process when treating feeds containing less than 20 pct near gravity material. It is just as efficient as the Chance cone but not so efficient as the Tromp bath when treating feeds containing from 20 to 50 pct near gravity material, and is slightly more effective than the Chance cone but not so effective as the Tromp bath when treating feeds containing in excess of 50 pct near gravity material.

The Ridley-Scholes dense-media bath is the least effective of the dense-media separators.

The two classifiers, the Menzies cone and the Hydroseparator, are less effective than the Ridley-Scholes bath and considerably less effective than the Tromp, Chance, or Link-Belt equipment. When com-

Table II. Separating Gravity Data for a Variety of Coal Cleaning Equipment

Cleaning Unit: Type	Cleaning Unit: Make	Range of Gravities Covered by Test Data
Sand media	Chance cone	1.40 to 1.45
Bone media jig	Jeffrey-Baum	1.40 to 1.60
Magnetite media	Link-Belt drum	1.35 to 1.732
Magnetite media	Tromp bath	1.35 to 1.835
Pulsator jig	Elmore	1.50 to 1.70
Pulsator jig	Jeffrey diaphragm	1.60 to 1.70
Pulsator jig	Vissac	1.45 to 1.60
Classifier	R & S & Menzies hydroseparator	1.44 to 1.70
Classifier	Menzies cone	1.42 to 1.70
Classifier	R & S hydrotator	1.30 to 1.70
Wet table	Deister Concenco No. 7	1.35 to 1.60
Air table	Stump	1.40 to 1.70
Air table	American	1.40 to 1.60
Magnetite media	Ridley-Scholes	1.45 to 1.60
Magnetite and misc. media	Dutch cyclone	1.35 to 1.80
Spiral	Humphreys	1.38 to 1.60
Wet table	Deister Machine Plat-O	1.38 to 1.60

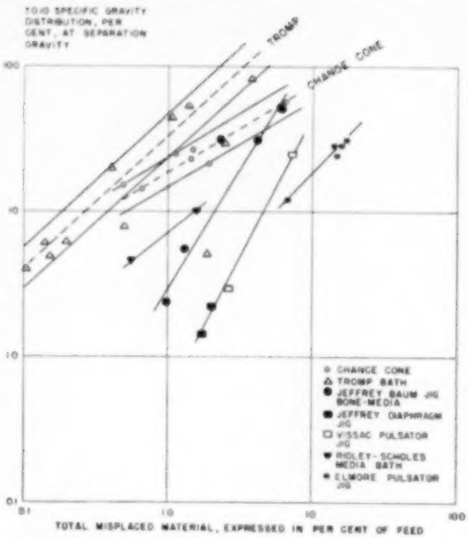

Fig. 1—Comparative effectiveness of coarse coal cleaning equipment.

pared with each other, the Menzies cone is more effective than the Hydroseparator.

The Jeffrey diaphragm and Vissac pulsator jig data plot on essentially the same line and are so plotted, although there undoubtedly will be some objections to such a procedure. When treating coals containing small amounts of near gravity material, the slow-speed pulsator jigs are less effective than the Hydroseparator. The jigs become progressively more effective as the amount of near gravity material increases, however, and in the range of 25 pct ±0.1 near gravity material they are just as effective as the Menzies cone. Extrapolation of the slow-speed pulsator jig data indicates that in the range of 80 to 90 pct ±0.1 the jigs would be just as effective as the Chance cone.

The Elmore high-speed pulsator jig is more effective than the Hydroseparator but less effective than the Menzies cone. Extrapolation of Elmore data indicates that in the range of 0 to 7 pct ±0.1 near gravity material the Elmore jig is more effective than the slow-speed Jeffrey and Vissac pulsator jigs.

The Jeffrey-Baum jig, and it should be emphasized that all of the test data are for jigs operated with a bone-media, is not so effective as the Menzies cone when treating coals containing from 0 to 3 pct ±0.1 near gravity material but is at least as effective and possibly more effective than dense-media equipment when treating coals containing 50 to 60 pct ±0.1 material. To a certain extent this conclusion is obscured by two factors. In every case the jigs were making a middling product which was being crushed, and either a portion or all of the crushed material was being returned to the jig feed. Since the difficulty of the separation or the percentage of ±0.1 near gravity material was determined from the raw coal float-and-sink data, the correlations obtained are bound to be somewhat in error. On the other hand, in two cases the Jeffrey-Baum jig was being used to treat all sizes down to zero, while all of the dense-media processes stopped short at some fairly coarse limiting size.

Intermediate and Fine Coal Cleaning Units: Fig. 3 presents in graphic form correlations developed for the intermediate and fine coal cleaning units or units used to treat: 1—coals having top sizes between ½ and ⅛ in., with no limitation on the bottom size, and 2—coals having top sizes of less than ⅛ in. The intermediate units for which data were available include 1—the Roberts & Schaefer Hydrotator, 2—the Deister-Concenco No. 7 wet table, 3—the Stump air table, 4—the American air table, 5—the Dutch cyclone dense-media process, both with and without the addition of a stabilizing material, and 6—the Deister Machine Co. Plat-O wet table. The only fine coal unit for which data were available is the Humphrey's spiral.

The Dutch cyclone using a stabilized media is the most effective separator in the intermediate group. Removal and control of the stabilizing material, whether Loess or Bentonite, is reported to require careful spraying and thickening steps and may require the use of froth flotation.

The Dutch cyclone using magnetite, roasted pyrites, or barytes without the addition of a stabilizing agent, the Deister Concenco No. 7 wet table, and the Roberts & Schaefer Hydrotator are all equally effective but are less effective than the Dutch cyclone operated with a stabilized media. It should be pointed out that the Dutch cyclone and Hydrotator tests were made on feeds sized or classified between 16 and 48 mesh, while the Deister table tests were made on relatively unsized feeds.

It might be argued that only the data for the Dutch cyclone and the No. 7 wet table cover the entire range of the correlation and that an unwarranted extrapolation is being made for the Hydrotator. However, in the range of separations for coals having small amounts of ± 0.1 near gravity material the three units are comparable and the

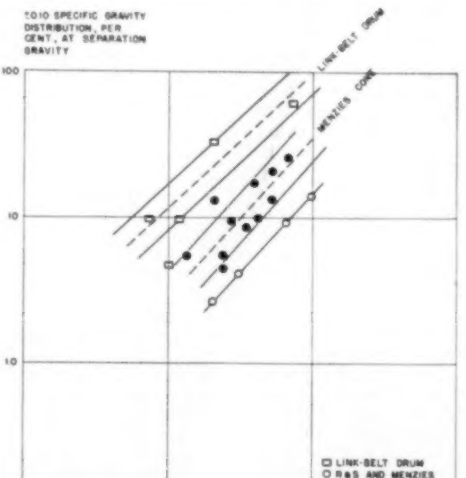

Fig. 2—Comparative effectiveness of coarse coal cleaning equipment.

Table III. Capacity Data

Test No.	Clean Coal Produced Tons Per Hr Per Sq Ft	±10 Distribution Pct at Gravity of Separation	Cleaning Unit Tested
1	0.68	57.5	Chance cone
3	0.85	22.7	Chance cone
4	0.90	14.7	Chance cone
5	1.68	15.5	Chance cone
6		25.0	Chance cone
7	1.37	27.2	Chance cone
8	1.60	55.0	Jeffrey-Baum jig
9	1.60	31.0	Jeffrey-Baum jig
10	1.60	33.0	Jeffrey-Baum jig
11	2.17	33.0	Link-Belt drum
14	0.91	5.3	Tromp separator
15	0.97	6.22	Tromp separator
16	0.87	30.20	Tromp separator
17	0.92	6.40	Tromp separator
18	0.27	29.60	Elmore jig
19	0.28	29.60	Elmore jig
22	1.89	14.30	Menzies hydroseparator
23	2.62	2.60	Menzies hydroseparator
27	1.78	17.50	Menzies cone
28	1.45	26.00	Menzies cone
29	2.28	5.50	Menzies cone
32	1.50	4.00	R & S hydrotator
33	1.13	2.96	R & S hydrotator
34	0.86	5.80	R & S hydrotator
54	5.46	1.54	Dutch cyclone
55	6.81	2.63	Dutch cyclone
56	7.34	14.43	Dutch cyclone
57	9.10	19.20	Dutch cyclone
58	8.40	47.50	Dutch cyclone
59	7.82	32.60	Dutch cyclone
60	8.10	81.00	Dutch cyclone
61	6.38	14.00	Dutch cyclone
62	7.82	8.70	Dutch cyclone
63	1.94	72.60	Dtuch cyclone
64	1.73	72.60	Dutch cyclone
65	2.70	70.20	Dutch cyclone
66	1.51	61.00	Dutch cyclone
67	1.09	78.60	Dutch cyclone
68	1.57	76.60	Dutch cyclone
69	1.31	71.90	Dutch cyclone
70	2.44	74.40	Dutch cyclone
71	1.85	75.70	Dutch cyclone
78	0.10	27.80	Deister Plat-O
79	0.10	6.53	Deister Plat-O
80	0.11	21.90	Deister Plat-O
81	0.11	3.60	Deister Plat-O
83	2.15	2.40	Jeffrey-Baum jig
84	2.97	25.00	Vissac pulsator jig
85	3.53	3.00	Vissac pulsator jig
94	0.28	25.00	Elmore jig
95	0.42	12.50	Elmore jig
96	0.26	31.40	Elmore jig
97	3.67	9.70	Link-Belt drum
101	0.48	20.00	Tromp separator
102	0.57	8.00	Tromp separator
103	0.58	84.00	Tromp separator
104	0.38	55.00	Tromp separator
105	0.47	46.50	Tromp separator

definite straight line correlations obtained for all of the sets of data indicate that the assumption is reasonable.

The Deister Machine Co. Plat-O table and some of the data for the Deister-Concenco No. 7 table show less effective separations than those for the Dutch cyclones, Hydrotators, and the bulk of the No. 7 tables. The variations of the No. 7 table can be explained as being inefficient operations and the lesser effectiveness of the Plat-O table could be explained as being inherent in the machine. However, it should be mentioned that the Plat-O data, while representing recent tests, are for old machines operated under irregular raw coal kind and size-consist conditions.

The air tables, and it should be noted that all of the tables were operated with a middling recirculation, are definitely the least effective of the intermediate group when treating coals containing small amounts of near gravity material. On the other hand, when treating coals containing between 30 and 45 per cent ± 0.1 near gravity material, the air tables are just as effective as the Plat-O table. Extrapolation of the air table data indicates that in the range of 70 to 90 per cent ± 0.1 material the air tables would be almost as effective as the Dutch cyclone, No. 7 table, Hydrotator group. The American table is slightly more effective than the Stump table in the lower ± 0.1 range but shows about the same effectiveness in the higher ± 0.1 range.

The only fine coal cleaning unit correlation shown in Fig. 3 is for the Humphrey's spiral treating 8-mesh × 0 or 8x100 mesh feeds. Since the Humphrey's spiral is alone in its class, as far as available operating data and correlations are concerned, it cannot be evaluated on a comparative basis.

Capacity Comparisons

An operating man is frequently more interested in the capacity of a unit than he is in its separating effectiveness and is willing to sacrifice effectiveness to some extent if by so doing he can materially increase capacity.

A line of reasoning similar to that presented under the section on the Correlation Theory suggested that there should be a correlation between the difficulty of a separation, or the amount of ± 0.1 material in the feed at the separating gravity, and the capacity for any make of coal cleaning equipment. Capacity was considered to be the tons of clean coal produced per hour per square foot of operating surface.

Table III contains a summary of the data available. Column 2 of Table III may be considered column 13 in Table I, thus eliminating duplications between Tables I and III.

Coarse Coal Cleaning Units: Fig. 4 presents in graphic form the capacity correlations obtained for coarse coal cleaning units, or units used to treat coals having top sizes in excess of ½ in. with no limitation on the bottom size. The coarse coal units for which data were available include 1—the Tromp process, 2—the Chance cone, 3—the Link-Belt drum, 4—the Jeffrey-Baum jig operated with a bone recirculation, 5—the Vissac pulsator jig, 6—the Menzies cone, 7—the Menzies Hydroseparator, and 8—the Elmore jig. Some of the relations show con-

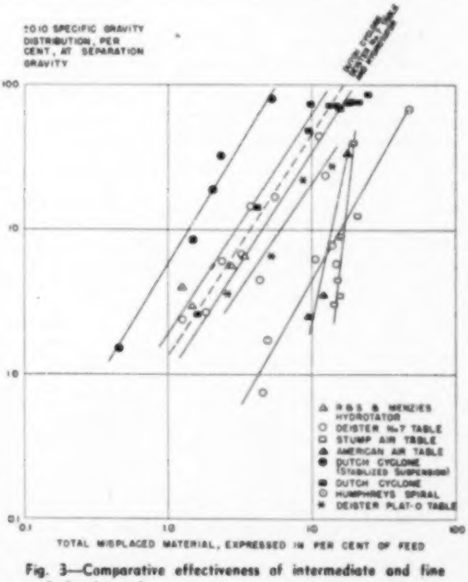

Fig. 3—Comparative effectiveness of intermediate and fine coal cleaning units.

siderable variation from an average condition; others are based upon only two or three points and are possibly in error, but all the data show trends and can be used for first approximation purposes.

The Vissac pulsator jig and the Link-Belt heavy-media drum have the greatest capacity per unit area of all the coarse coal cleaning units. The Vissac jig was used to treat sized feeds, and its high capacity can be explained on that basis.

The Jeffrey-Baum jig can produce just as much washed coal per unit area as the Link-Belt drum when treating coals containing 60 pct or more of near gravity material but has a relatively lower and lower capacity rating as the difficulty of the separation decreases.

So similar are capacity relationships of the Menzies cone and the Menzies Hydroseparator that they have been combined and plotted on the same line. The Menzies combination shows a lower capacity rating than the Jeffrey jig when treating coals containing more than 15 pct near gravity material, and a higher capacity rating than the Jeffrey jig, but not so high a capacity as the Link-Belt drum, when treating coals containing less than 15 pct near gravity material.

The Chance cone shows considerably less capacity than the Jeffrey jig or Menzies combination at difficult separations but a greater capacity than either when treating less difficult coals.

The Tromp process has the lowest capacity of all the heavy-density equipment. The Elmore jig has the lowest capacity of all of the coarse coal equipment for which data were available.

Intermediate Coal Cleaning Units: Fig. 5 presents capacity correlations obtained for intermediate coal cleaning units, or units used to treat coals with top sizes between ½ and ⅛ in., with no limitation on the bottom size. The intermediate coal units for which data were available are 1—the Roberts & Schaefer Hydrotator, 2—the Dutch cyclone dense-media process, and 3—the Deister Machine Co. Plat-O wet table. Paucity of available data leaves room for misrepresentation, but again, all the data show definite trends.

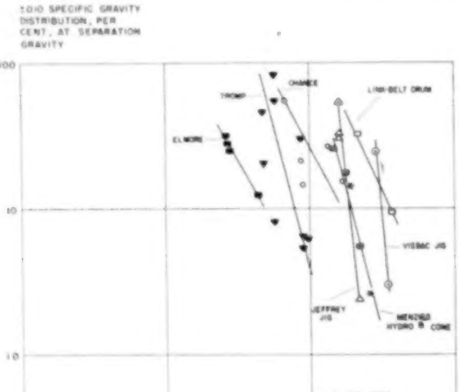

Fig. 4—Relationship between difficulty of separation and unit capacity for coarse coal cleaning units.

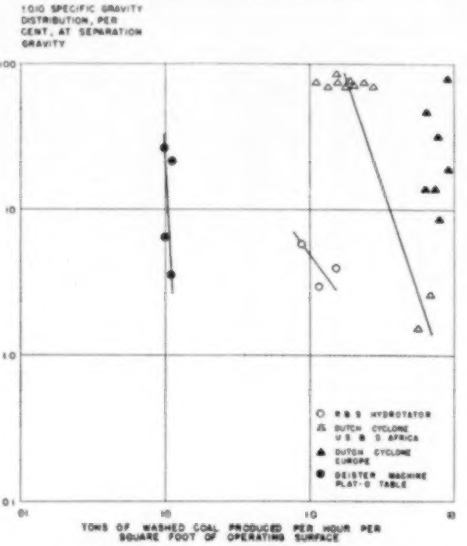

Fig. 5—Relationship between difficulty of separation and unit capacity for intermediate coal cleaning units.

The Dutch cyclone heavy-media process has tremendous capacity per unit area when compared with the R & S Hydrotator and the Deister Machine Co. Plat-O table. In this connection it is interesting that differences in capacity are reported by United States and South African investigators on the one hand and European investigators on the other. The European investigators reported considerably higher capacities and the values were approximately the same irrespective of the difficulty of the separation.

Conclusions

Conclusion No. 1: The total amount of misplaced material in washed coal and refuse products, as determined at approximate gravity of separation and expressed as percent of feed, is proportional to the percentage of ±0.1 near gravity material in the feed.

Each type and make of cleaning equipment has its own particular relationship, and to develop the separation characteristics of any particular piece of equipment it is necessary to obtain test data for several installations or for several tests at different separation gravities at the same installation.

Data for individual size fractions in composite feed may be used for correlation purposes, and will plot in the same way as composite size-consist data, if float-and-sink data are available for that particular size in the raw coal, washed coal, and refuse.

Data for both anthracite and bituminous coals provide the same relationships. The required specific gravity of the separation and the particle shape, i.e., the presence of thin, flat slate, may prevent the use of some piece of equipment for a specific washing job, but within the operating limits of the equip-

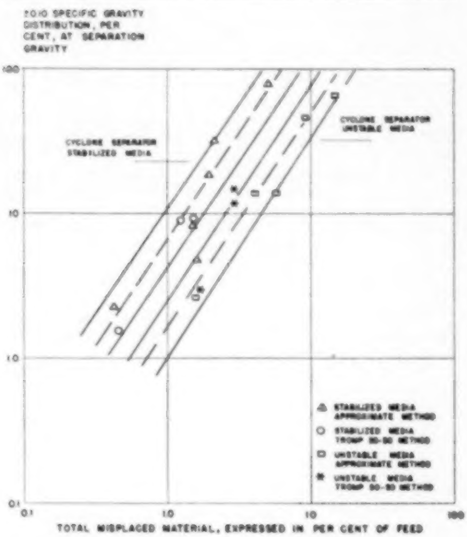

Fig. 6—A comparison of the correlations obtained when using the Tromp 50-50 method and the approximate method of determining gravity of separation.

ment, the specific gravity of the material has no influence on the correlations obtained.

To place all the data on a comparable basis, it is necessary to adjust the percentage of ±0.1 near gravity material to allow for the amount of sink 2.0 sp gr material in the feed.

Conclusion No. 2: To determine the effectiveness of a particular washing unit, it is necessary to have complete float-and-sink data for the composite raw coal. It should be emphasized that this data need be only for the composite size actually treated and not for each individual size fraction. It is not necessary to conduct exhaustive float-and-sink tests on the products of the washing unit. A one-gravity separation is all that need be used if it is approximately the gravity of the separation. However, it is preferable to conduct tests at a number of gravities to determine the separation gravity by the 50-50 method described by Yancey and Geer in 1938.[a]

Conclusion No. 3: The total amount of misplaced material, as determined at the approximate gravity of separation, expressed in percent of the feed, and representing a particular separation by a particular piece of equipment, may be distributed in varying proportions between washed coal and refuse, but its total remains essentially constant. Thus washing any coal with its constantly changing size consist and physical make-up in a given type and make of cleaning equipment at a given and approximately constant separation gravity normally results in a washed coal containing varying amounts of misplaced material; in consequence, the ash content of the washed coal will show considerable variance. It seems implied that regulation of the equipment used will allow more or less misplaced material to remain in the washed coal and that it is possible, speaking relatively, to obtain either a high ash or low ash washed coal while making a separation at essentially the same specific gravity.

Conclusion No. 4: The capacity of any given coal cleaning unit, when capacity is expressed as the tons of clean coal produced per hour per square foot of operating surface, is related to the percentage of ±0.1 near gravity material in the feed, or the difficulty of the separation.

Summary

The relative effectiveness ratings of most of the equipment discussed in this paper will not surprise any real student of coal preparation. Most preparation men have already independently arrived at the same or nearly the same conclusions by less direct methods of comparison. The misplaced material correlations merely corroborate conclusions already industry-wide, but do so on an exact basis. The capacity correlations likewise will surprise very few people. The correlations do provide reference points for future work, and use of the method will allow the evaluation of other makes and types of coal-cleaning equipment, both old and new.

The information presented in this paper indicates that certain machines are less effective separators of coal from refuse than others, and that some machines are capable of producing more tons of clean coal per hour per square foot of operating surface than others. This should not be construed as re-

Table IV. Cyclone Separator Data

Test No.	Media	Washed Coal, Wt, Pct	Washed Coal, Sink, Pct	Refuse, Wt, Pct	Refuse, Float, Pct	Gravity of Separation, 50-50 Method	Gravity of Separation, Approx. Method	±0.1 Distribution at Gravity of Separation	Total Misplaced Material, Pct of Raw Coal
54	Stabilized	95.70	0.20	4.30	6.25	1.63		1.54	0.46
54	Stabilized	95.70	1.70	4.30	0.60		1.50	4.93	1.65
54	Stabilized	95.70	0.30	4.30	3.40		1.60	2.36	0.43
55		88.92	1.30	11.08	5.00	1.54		2.98	1.71
55		88.92	1.12	11.08	5.47		1.55	2.63	1.60
56		33.30	7.50	66.70	0.80	1.52		15.00	3.03
56		33.30	4.38	66.70	4.10		1.80	14.43	4.19
57	Stabilized	86.25	2.50	13.75	6.50	1.49		9.25	1.32
57	Stabilized	86.25	1.90	13.75	2.70		1.45	19.20	2.09
58		77.20	1.80	22.80	7.30	1.79		15.30	3.10
58		77.20	9.99	22.80	8.01		1.70	47.50	9.54
59	Stabilized	88.30	0.80	11.70	7.20	1.505		9.20	1.55
59	Stabilized	88.30	2.28	11.70	2.59		1.45	32.00	2.31
57*	Stabilized	86.25	5.78	13.75	1.34		1.40	81.00	5.17
58†		77.20	1.19	22.80	21.76		1.80	14.00	3.88
59**	Stabilized	88.30	0.82	11.70	6.85		1.50	8.70	1.52
63		41.30	20.30	58.70	10.90	1.35	1.30	72.60	14.78
65		60.00	20.30	40.00	10.89	1.25		71.20	16.54
65		60.00	13.40	40.00	17.30		1.40	70.20	14.96

* Test 60 in original text.
† Test 61 in original text.
** Test 62 in original text.

flecting on the usefulness of some of the equipment. First cost and operating costs have not been taken into account in this evaluation, and since performance is only one factor among many in the selection of coal-cleaning equipment, other factors may outweigh the limitations of a slightly less efficient separation or a lower capacity.

Appendix

The term *gravity of separation* as used in this paper embraces two concepts for comparison purposes. The first concept is the one developed by Tromp in which *gravity of separation* is that gravity at which the distribution is 50 pct to clean coal and 50 pct to refuse. This is the only method the author would use if he wished to determine the gravity at which a separation is being made for a particular piece of equipment in a particular preparation plant. The second concept is one developed by the author in which *gravity of separation* is an approximate value as determined by scanning the float-and-sink data for washed coal and refuse produced by a particular piece of equipment. One or two such approximate values may be determined from a particular set of data. For example:

Specific Gravity	Washed Coal Wt. Pct	Refuse Wt. Pct
Float 1.30	30.0	1.0
1.30 x 1.35	45.0	1.0
1.35 x 1.40	22.0	2.0
1.40 x 1.50	2.0	3.0
1.50 x 1.60	1.0	8.0
1.60 x 1.70		35.0
Sink 1.70		35.0

With these data, approximate separations are indicated at 1.40 and 1.50 sp gr. These gravities and their corresponding amounts of misplaced material and percentage of near gravity material cannot be used to determine the actual gravity at which a separation is being made but may be used for comparison purposes, since either concept will provide essentially the same correlation between the amount of ±0.1 near gravity material and the total amount of misplaced material.

Table IV contains data for cyclone heavy-media separators in which the gravity of separation has been determined by the Tromp 50-50 method. The corresponding amounts of near gravity material and misplaced material are included. Table IV also contains data for the same tests in which the gravity of separation has been determined by the approximate method, only one gravity being indicated in some of the tests and two gravities in others. Corresponding amounts of near gravity material and misplaced material are also included. Fig. 6 shows a plot of these values, which should be compared to the initial plot shown in Fig. 3. Both sets of values provide essentially the same correlation relationship.

The bulk of the values listed under Gravity of Separation in Table I were determined by the approximate method. The method is rapid, presents two values at times when the Tromp method presents only one, and provides good values for comparison purposes.

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Solids Fluidization Applied to Lime Burning

by F. S. White and E. L. Kinsella

Solids fluidization utilized in two ways for the commercial production of lime is described. Crushed —6 mesh limestone is dried and dedusted in a single bed reactor, then calcined in a 5-stage reactor. Construction, operation, and results obtained with both processes are given.

THE solids fluidization process brought out by the Standard Oil Development Co. in the early forties for catalytic cracking of petroleum enabled rapid transfer of large quantities of heat from gases to solids under closely controlled temperature conditions. As the technique developed, it was natural that applications for the process outside the petroleum industry would be sought. The calcination of limestone to quicklime requires the transfer of a large quantity of heat at high and closely controlled temperatures. For these reasons, this possible application was one of the first to be investigated.

Laboratory work conducted by the Dorr Co. indicated that lime produced by solids fluidization would be of exceptional quality. Calculation of theoretical heat balances showed that savings in fuel could be expected. Accordingly, a pilot plant of 10 tons per day capacity was erected by the New England Lime Co. at Adams, Mass., to substantiate the process. Operation of the pilot plant, previously described,[1] showed that excellent lime could be produced with a saving in fuel. The work also indicated the necessity of preparing feed to remove the finer fractions if dust losses were to be held to a tolerable level. A single-stage fluidized solids drier-stripper was developed to accomplish this sizing.[2] Subsequently a commercial lime reactor was erected at Adams, beginning operation in June 1949, followed by air sizer facilities completed in November 1951. This paper will describe these two applications of solids fluidization.

The raw material for preparation of lime by the fluidizing process is a high calcium crystalline limestone having the approximate chemical composition shown in Table I. The stone is first prepared by crushing and grinding in conventional equipment to pass a 6-mesh sieve. After crushing, the material is sent to the FluoDry unit for drying and dedusting.

Fluidized Drying and Sizing

As shown in Fig. 1, the FluoDry unit consists of a ¼-in. cylindrical steel shell, 9-ft ID by 22-ft 6-in. overall height. Attached horizontally at the bottom is a 6-ft diam combustion chamber, approximately 7 ft 6 in. long. A refractory constriction dome, consisting of first-quality fire brick shapes, partitions the inside into two sections. The lower section is called the windbox and the upper the bed compartment. The vertical wall of the shell is lined with 6-in. first-quality rotary kiln blocks. In the bottom of the windbox a hoppered section is formed with castable refractory to permit the windbox to be readily cleaned via a 6-in. screw conveyor. The combustion chamber is similarly lined with refractory shapes. On the 6-in. diam section, 4½ in. of insulating brick, HW-26, are used. The inner combustion chamber is lined with 4½-in. Korundal brick to withstand the high flame temperatures encountered.

An 8-in. steel feed pipe, entering vertically through the top and fitted with an air-operated cone valve,

F. S. WHITE, Assistant to the President, and E. L. KINSELLA are with New England Lime Co., Adams, Mass.

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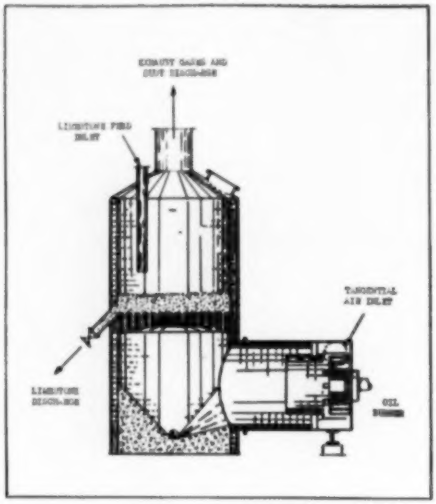

Fig. 1—Sectional view of FluoDry unit.

provides a means of introducing feed to the unit. The discharge consists of a 6-in. pipe at the top of the dome, running angularly to the outside and likewise fitted with an air-operated slide valve.

Air for fluidization and atomizing of the oil is supplied by two blowers. Primary air is supplied by a Buffalo blower rated 12,000 cfm at 2.0 lb gage and

Table I. Approximate Analysis of Limestone Feed

Composition	Pct
SiO_2	1.2 to 2.5
R_2O_3	0.3 to 0.5
$CaCO_3$	95.0 to 97.0
$MgCO_3$	1.0 to 1.5

Table II. Operating Data, FluoDry Unit

Bed temperature	200° to 225°F
Air flow	12000 cfm
Feed rate	100 to 125 tons per hr
Oil required	50 to 60 gal per 100 tons limestone feed
Power required	98 kw-hr per 100 tons limestone feed
Split	100 lb feed = 86 lb coarse feed plus 14 lb fines

driven by a 150-hp motor. Air for the oil burner is provided by a Buffalo blower rated 600 cfm at 2 lb gage. Exhaust gases from the sizer are passed through a series-parallel combination of four cyclone collectors, then through a continuously cleaned felt bag collector of the Hersey type.

Control of the drying temperature is effected by a recording temperature controller sensing the temperature via a thermocouple immersed in the fluidized bed. The instrument regulates the flow of oil by positioning the oil burner valve with a diaphram motor valve. The bed level is similarly regulated by a recording controller measuring the differential pressure of the fluidized bed. The instrument positions an air-operated slide valve to allow material to discharge at a rate which will hold a constant bed level. Indicating instruments are also utilized to measure pressure drops throughout the system and temperatures in the combustion chamber and dust collectors. Safety devices to guard against flame and power failures have been incorporated.

The sizer is operated in conjunction with the crushing plant, the screen discharge being the feed to the sizer. The operation of the sizer from start to shutdown is simple. After the blowers are started, the oil burner, using Bunker C oil, is lighted by an electrically ignited gas torch. The inlet valve is then opened, permitting feed to enter. Finally, the discharge valve is opened when the bed reaches the desired level. When in balance the unit is switched to automatic control, the operator then monitoring the unit from the control panel. When it is not in operation, the feed and the burner are turned off, the bed drained out and the blowers shut down. While company experience with this FluoDry unit has thus far been limited, there has been enough operating time under winter conditions to make possible a compilation of data, shown in Table II, and to justify the following tentative conclusions.

1—Mechanical operation of the unit is simple, and very low maintenance costs are anticipated. 2—It is an efficient drier, capable of drying limestone with a moisture content of 2 to 3 pct while burning 60 gal of Bunker C fuel oil per 100 tons limestone. 3—It is, however, a less efficient sizer than desired. Inspection of Fig. 2, a plot of the screen analyses of the feed, product, and dust, will show that the sized product still contains about 7.0 pct −100 mesh. This represents about 1/3 of the original −100 mesh fraction fed to the unit. While an improvement has been noticed in operation of the rotary kilns on this quality-sized and dried feed, less of an improvement has been found in operation of the FluoSolids kiln. For the latter, it would be highly desirable to remove all of the −100 mesh fraction.

Fluidized Calcination

The crushed and sized limestone is next processed to lime in the FluoSolids reactor. In Fig. 3, a sectional view, the reactor is seen to consist of a vertical steel shell 13 ft 6 in. ID by 45 ft high, closed at the bottom by a windbox having a perforated steel constriction plate, and at the top by an insulated steel cover with a gas take-off to the dust collectors. The reactor is lined throughout with 9 in. of 2600°F

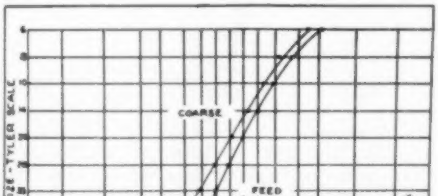

Fig. 2—Plot of screen analyses of FluoDry samples.

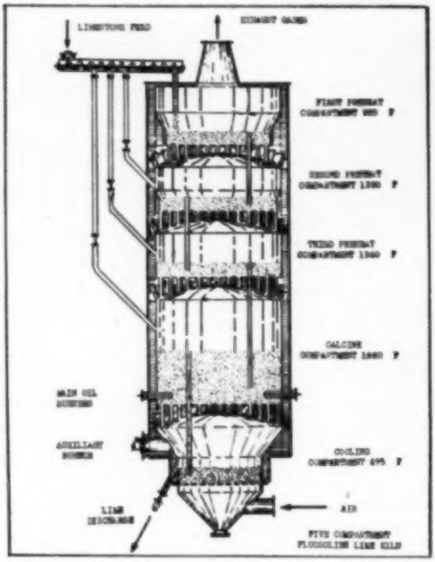

Fig. 3—Sectional view of FluoSolids kiln.

series insulating firebrick. Four firebrick constriction domes divide the unit into five stages, the top three being devoted to preheating incoming stone, the fourth to calcination, and the fifth to cooling of the lime.

The firebrick domes are pierced with holes approximately 3 in. in diam for upward flow of gases. The upper ends of these holes are fitted with stainless steel orifices drilled with smaller holes of varying sizes adjusted to give a pressure drop which will insure fluidization across the bed. Each dome is also fitted with an alloy steel pipe for transferring solids downward to the next successive stage. Fig. 4 is a photograph of the interior of the calcining compartment showing some of these details.

The reactor in operation can best be visualized from a description of the startup procedure. With the reactor empty, the normal air blast is started followed by ignition of the auxiliary burner firing into the cooling compartment. After a period of preheating, limestone is fed directly to the calcining chamber via a "side feed" pipe until a 3½ to 4-ft bed of fluidized stone is established. Feed is stopped, allowing the temperature of the bed to increase slowly to 800°F, whereupon the regular burners are inserted into the calcining bed.

These burners, 12 in number, are lengths of extra heavy carbon steel pipe which are inserted through packing glands into an insulated, air-purged burner jacket projecting radially 15 in. into the bed at a height of 6 in. off the constriction plate. A positive displacement pump driven from a common shaft by a variable speed motor supplies each burner with 1/12 the total Bunker C oil flow. No effort is made to atomize the oil with air or to preheat it beyond pumping consistency.

When the regular burners are in operation, the auxiliary burner is shut down. Combustion then transfers entirely to the fluidized bed, rapidly raising its temperature until 1540°F is reached. At this point calcination starts, the temperature then rising only slightly until calcination is completed. While the calciner bed is being reduced to lime, the three preheat beds are successively established. By the time the top bed has been made, calcination is usually completed, the temperature once again rising rapidly. A flow of feed sufficient to stabilize the temperature at 1860°F is then started into the first preheater. As each preheater is filled to the level of the top of its transfer pipe, the incoming feed starts a flow of displaced material through the stages to the calciner. Thus new feed is preheated to temperatures of 925°, 1360° and 1560°F, utilizing the heat in the combustion gases and the CO_2 evolved from the calcining stone.

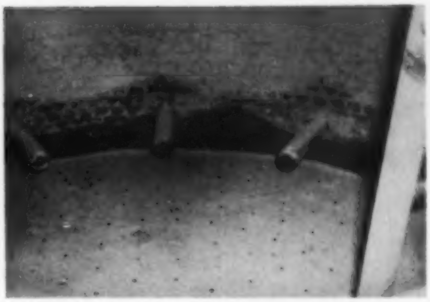

Fig. 4—Interior of calcining compartment.

Finally, when the calciner is filled to the overflow point, finished lime is allowed to flow to the cooler where the incoming air cools the lime to 700°F, and is itself preheated. Finished lime then discharges from the kiln through an automatically controlled slide valve similar to that previously described for the drier.

As shown in Fig. 5, the kiln is well instrumented. Production is controlled by manually setting the potentiometer which governs the speed of the oil pump motor. Air flow is also regulated manually to hold 3 to 5 pct excess air as determined by a recording oxygen instrument analyzing the stack gases. There are, in addition to the controlling instruments, indicating instruments which measure bed and constriction plate pressure drops, bed temperatures, and back pressures on the oil guns to warn of plugging.

With constant heat input the amount of limestone feed required would be uniform with homogenous feed. However, variations caused by segregation in the storage bins and varying chemical compositions

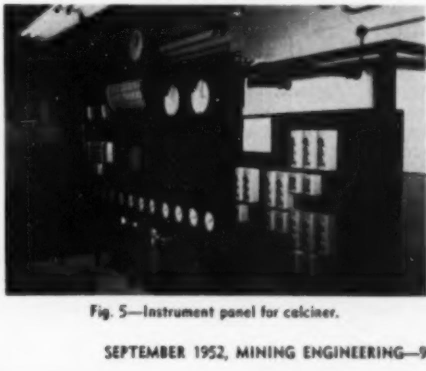

Fig. 5—Instrument panel for calciner.

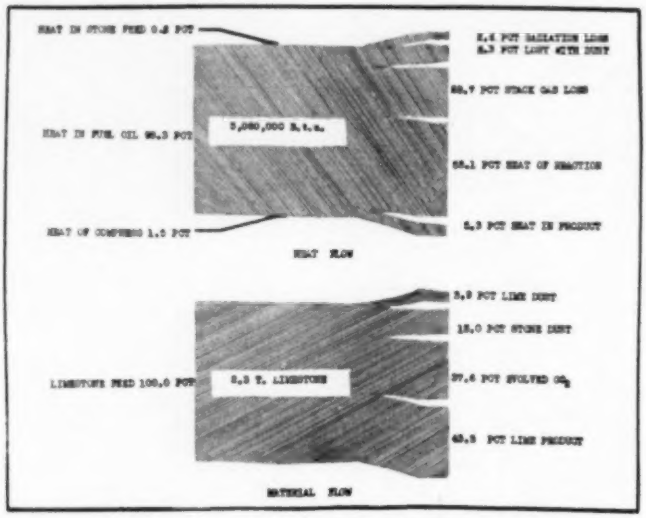

Fig. 6—Heat and material flow diagrams based on one ton of product.

cause temperature fluctuations. These are detected by a recording controller which measures the calciner temperature and then varies the speed of a constant weight belt feeder. Except for serious upsets, temperatures are controlled to ± 20°F of the desired values. Over a period of time ratios of stone fed to oil burned have been surprisingly constant. Typical data for a 24-hr period of operation are presented in Table III.

Table III. Data for Heat and Material Balances

Oil consumption	2907 gal per 24 hr
Oil temperature	160°F
Oil analysis	87 pct carbon
	10 pct hydrogen
	152,000 Btu per gal, gross
Lime produced	89.0 tons
Lime temperature	690°F
Stone used	204.0 tons
Stone temperature	70°F
Air flow	4,850,000 cf (stp)
Air temperature after compression	150°F
Stack gas temperature	900°F
Stack gas analysis	34.0 pct CO_2
	1.0 pct O_2
Dust analysis	32.0 pct lime
	68.0 pct limestone

By means of this data heat and material balances were calculated, the results being graphically shown in Fig. 6. Inspection of these balances will reveal a major advantage and a disadvantage of the fluidized process for burning lime.

The major advantage of the process is in the fuel economy. Approximately 5 million Btu of heat are required to produce one ton of recoverable lime. This figure compares favorably with a well-run modern shaft kiln and is lower than that required for a modern rotary kiln, where 7 to 8 million Btu will be used. In the New England area where fuel costs are high this is an important saving.

On the debit side is the fact that yield of lime from stone is only about 76 pct of theoretical, as compared to 100 pct in a shaft kiln and 90 pct or thereabouts in a rotary kiln. Recoveries obtained in the 3-stage pilot furnace, with feed quality only slightly better than that currently being processed, were of the order of 90 pct, which would be considered satisfactory. Reason for greater dust loss in the commercial unit has not yet been determined.

Quality of the fluidized product is excellent. Repeated analyses of the product show the residual CO_2 contents to be less than the 0.15 pct representing 99.8 pct removal of CO_2. To obtain this degree of calcination in a rotary or shaft kiln would require operating temperatures in excess of 2000°F, which would tend to flux the impurities with the CaO, thus lowering the available lime content of the finished product. Fluxing is not appreciable in the temperature range of 1800° to 1900°F employed in the fluidized process. As a result the available lime contents obtained in the fluidized product approach very closely those predicated by the stone analyses.

Reactivity of the lime with water can be made to vary by changing the burning temperature. Whereas the normal product has a reaction speed of 60 sec when slaked with 2½ parts water at 100°F, the rate can be advanced or retarded through a range of 25 to 400 sec by lowering or raising the temperatures from the normal operating point of 1860°F.

In summation, the first attempts to apply fluidized solids technique to the manufacture of lime burning have resulted in a new method for drying and sizing limestone on a large scale and an efficient method for calcining limestone to a quicklime of high quality.

Acknowledgments

We are indebted to the engineers of the Standard Oil Development Co. and the Dorr Co. who developed the fluidization technique, to C. C. Loomis of New England Lime Co., who encouraged these particular applications, and to W. J. Barrett for preparing the photographs.

References

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aime NEWS

Administrative Matters Delegated to Branches

Considerable discussion took place at the June 18th Board of Directors Meeting concerning ways the Petroleum Branch office in Dallas could better serve its members and the essentially Petroleum Local Sections, namely the Delta, East Texas, Gulf Coast, Kansas, Mid-Continent, North Texas, Oklahoma City, Permian Basin, and Southwest Texas. The question of whether decentralization strengthens or weakens the Institute was debated, particularly as to what services could be advantageously decentralized and how both offices could keep informed on their mutual problems. Among other things, it was decided that an Auxiliary Admissions Committee could screen applications from Petroleum members before final action by the National Admissions Committee in New York. Finally it was voted, "That all administrative matters that can be properly delegated to a Branch organization may be so delegated". Final details were approved at the July 16th meeting of the Executive and Finance Committees.

Mineral Industries Convocation Symposium

The backbone of the Centennial of Engineering at Chicago, September 3 to 13, is the symposium program. Of interest to AIME members is the Minerals Industries portion of the program Monday and Tuesday, September 8 and 9.

The basic theme of the Centennial is enhancement of public appreciation of engineering and its contribution to the development of the U. S. and the advancement of civilization.

**Monday and Tuesday, September 8 and 9
Hotel Sherman Ballroom**

Chairman: Clyde Williams, Director, Battelle Memorial Institute

Secretary: Richard J. Anderson, Assistant Supervisor, Battelle Memorial Institute

Exploration for Metals, Petroleum and Water, William E. Wrather, Director, United States Geological Survey

Mining and Quarrying, Donald H. McLaughlin, President, Homestake Mining Company

Coal-mining, Preparation, and Utilization, J. B. Morrow, Consulting Mining Engineer, Alford, Morrow and Associates

Ore Beneficiation and Hydrometallurgy, O. C. Ralston, Chief Metallurgist, United States Bureau of Mines

Nonmetallic Materials, John D. Sullivan, Assistant Director, Battelle Memorial Institute

Iron and Steel Production and the Coke Industry, Walther Mathesius, Consultant, Freyn Engineering Division, Koppers Company, Inc.

Nonferrous Smelting and Refining, R. W. Diamond, President, Consolidated Mining and Smelting Company of Canada, Ltd.

Synergism of Engineering and Petroleum, Robert E. Wilson, Chairman of the Board, Standard Oil Company (Indiana)

The Role of Metals in our Economy, Zay Jeffries, Vice President, General Electric Company

Ask Coal Div. To Aid School Fund

The members of the Coal Division of AIME were circularized on May 5, 1952, for contributions to the Scholarship Fund. It was pointed out that additional gifts were needed in order to keep up the scholarships that had been granted to students in mining engineering at Colorado School of Mines; Lafayette College; Lehigh University; the Ohio State University; Pennsylvania State College; University of Pittsburgh; Virginia Polytechnic Institute; and West Virginia University. Up to the present time, the total amount is not sufficient to meet the obligations of the Fund during the next college year. Therefore, it is hoped that those who care to contribute will send checks promptly to the office of AIME in New York designating the gifts as contributions to the Scholarship Fund, Coal Division, AIME.

AIME Directors Appoint Official Representatives

At the June 18th meeting of the Board of Directors appointments of official AIME representatives were made as follows: Oliver C. Ralston was appointed to succeed himself as AIME representative on the div. of engineering and industrial research of the National Research Council for a three-year term expiring July 1, 1955. Maxwell Gensamer was appointed as AIME representative on the interim board of governors of *Acta Metallurgica* to be published next year and sponsored by the American Society for Metals in cooperation with several other societies. Michael L. Haider was appointed to succeed W. E. Wrather as AIME representative on the John Fritz Medal Board of Award of the four Founder Engineering Societies for a four-year term expiring Sept. 30, 1956. E. C. Meagher was appointed to succeed himself as AIME representative on United Engineering Trustees for a four-year term expiring October 1956. R. M. Dickey accepted the appointment as AIME representative on the American Standards Assn., Sectional Committee B30, Safety Code for Cranes, Derricks and Hoists. Stefan Boshkov was appointed to succeed W. H. Loerpabel, resigned, on the Technical Publications Committee, AIME, for the balance of the term expiring Oct. 15, 1952.

Local Sections Vie For President's Banner

The Local Section membership contest this year will be based on applications received at Institute headquarters on April 1 through December 31. Applications for change of status from Student Associate to a higher grade apply as credits but changes of status other than these cannot be credited.

The section showing the greatest percentage increase will be the winner of the President's Banner. A second award will go to the section submitting the greatest number of new applications.

THE DRIFT OF THINGS

by Edward H. Robie

New Name for the Institute?

SHOULD the name of our Institute be changed? Perhaps one should say, should it *again* be changed, for until 1919 the name was the American Institute of Mining Engineers. When the American Institute of Metals joined our ranks we became the American Institute of Mining and Metallurgical Engineers. To make it a full mouthful, and strictly legal, one should add the word "Incorporated."

At the June meeting of the Board of Directors it was suggested that this name was not a particularly happy one, especially so far as members of the Petroleum Branch, comprising a quarter of our membership, were concerned—not particularly appropriate as the name for the professional society for petroleum engineers. Apart from the petroleum group, the present name is felt by some not to be sufficiently comprehensive. Most geologists and many metallurgists, ceramists, and fuel technologists, for instance, are not engineers. They could never qualify as registered professional engineers under the licensing laws and yet they are valued and fully qualified Members of the AIME.

The Directors, at the June meeting, thought the idea of a change in name should be presented to the membership for discussion, and suggested this column as the avenue of approach.

Nine years ago Dean Steidle, of Penn State, recommended among other things that the name of the Institute be changed to "American Institute of Mineral Technologists and Engineers." In an editorial in *Mining and Metallurgy* in January 1944 objection was offered to this name but "American Institute of Mineral Engineers" was suggested. A special committee appointed by the Board to review Dean Steidle's suggestions (W. B. Heroy was Chairman) suggested that if the AIME were starting from scratch, the American Institute of Mineral Engineers might be a preferred name. It was not thought that the change suggested by Dean Steidle was one of major importance, and the comment was made that "the present name has gained prestige through many years and is cherished by our membership. An important element of 'good will' is attached to it and some of this would be unquestionably sacrificed by any change."

In *M&M* for February 1945, Oliver C. Ralston went into the derivation of the terms mineralogy and metallurgy, and pointed out that both Mining and Metallurgical mean "digging," etymologically. He went on to show, with tongue in cheek, that if we want a short and all-inclusive name it should be American Institute of Geo-urgists, or, for euphony, Geolurgists. Oliver's brilliant development seems to have been received with speechless wonder!

Should a decision ever be made to change the name, we think most members will agree that the American Institute of Mineral Engineers is to be preferred over anything yet suggested. "Mineral" has become, through usage, a much broader term than is indicated in the dictionary. Webster states that a mineral, in the true sense, must be inorganic, but Murray's Oxford English Dictionary seems more liberal, defining "mineral" as "any substance which is obtained by mining; a product of the bowels of the earth;" and "mineral oil" as any oil of mineral origin, as petroleum or shale oil. Certainly "the mineral industry" is ordinarily taken to include those activities based upon processing all the metallic and nonmetallic ores and deposits found in the ground, including the fuels. One advantage of this title is that the familiar abbreviation, "AIME" need not be changed. The advisability of maintaining a well-known abbreviation is evident, and was recognized, for instance, when the Committee for Industrial Organization became the Congress of Industrial Organizations some years ago.

But if the petroleum engineers are the ones to be chiefly considered in revising the name of the Institute, it is by no means certain that they would feel that they are much more adequately recognized in a group of Mineral Engineers than in a society of Mining and Metallurgical Engineers. And then there remain, of course, the considerable number of professional members of the Institute who are not engineers at all.

Comment from readers is invited.

Labors of Love

Few members of the Institute ever stop to think of the time, thought, and money that is given freely and generously by hundreds of members to their professional society. A large part of the President's year is devoted to traveling about the country for the society, to attending meetings, and to reviewing matters on which his counsel is needed. Most of the Directors, particularly those who are able to come to frequent meetings, also take their jobs seriously. Branch, Division, and Local Section officers, particularly Chairmen and Secretaries, find that many hours a month are required, which usually means that much less leisure time at home. Several hundred members serve on Institute committees, which may require scarcely any time or may require a day's work a month. Such committees as the Admissions Committee and the various Technical Publications committees are among the more active and hard-working groups. Many who are not on any committee still find they are called on for work. Such, for instance are those especially skilled in specific fields who are asked to review critically technical papers submitted for the Transactions.

It is true that most of those who thus contributed to the work of the Institute benefit themselves by so doing, by widening their knowledge and their circle of friends and acquaintances in the profession, but for most this is a byproduct of their work and not the reason for their acceptance of such professional responsibility. Many of our workers are at or near the top in their companies so added recognition from Institute work means little to them.

Undoubtedly no living member of the Institute has given more of his time to its affairs, or has been a more diligent, able, and devoted guardian of its material well-being, and supporter of its traditional purposes and tenets, than Erle V. Daveler, who retired from the Board last February after 23 full years of service as a Director. For nine years he was Vice-President. He served as Chairman of both the Executive and Finance Committees of the Institute, and is still Chairman of its Committee on Investments. Before coming to New York he served on professional committees and contributed to the Transactions.

Such men as Erle Daveler, and conspicuously he, have brought the AIME to its present stature.

At the suggestion of the Board, President Haider recently transmitted to him a letter of appreciation. In replying he said, "I have always considered it a deep privilege to serve the Institute, and through my association with it there have resulted friendships with the finest group of men I know of in any organization, who all unselfishly have served it."

Personals

A. A. Almstrom is mill superintendent for Barrue Mines, Barraute, Quebec.

Frank A. Ayer, vice president of Copper Range Co., who has been in charge of the development and bringing into production of White Pine orebody in northern Michigan, has resigned.

E. G. Bailey and **H. S. Mudd** were made honorary members of the American Society of Civil Engineers.

Lee Bilheimer, graduate of Missouri School of Mines, has been employed as assistant mine captain at the Edwards, N. Y., div. of the St. Joseph Lead Co.

Douglas C. Blackwell, recently graduated from the South Dakota School of Mines, is now with the Oliver Iron Mining Co. of Duluth, Minn.

Joseph F. Brown has joined the Grand Junction Exploration Branch of the United States Atomic Energy Commission as a geologist.

JAMES T. FINLEN

James T. Finlen, member of the legal staff of the Anaconda Copper Mining Co. since 1928, has been named western general counsel for the company. The position was vacated by **Roy H. Glover** who was elected vice president and general counsel.

Joseph C. Kieffer was appointed assistant manager of AS&R operations at Wallace, Idaho and not at Salt Lake City as previously mentioned.

R. D. Longyear returned home after a four months' trip through Africa, Europe, England and Scotland. Longyear, president of E. J. Longyear Co. presented a paper on "Trends in Diamond Drilling in the United States" at a diamond drilling symposium sponsored by the Chemical, Metallurgical, and Mining Society of South Africa, at Johannesburg.

Paavo V. Maijala, formerly superintendent, has been appointed chief mining engineer of the Outokumpu Company's Ylojarvi mine in Finland.

Lawrence Litchfield, Jr. has been appointed president of Alcoa Mining Co., succeeding **Frank B. Cuff.** Mr. Litchfield also succeeds Cuff as president of Surinaarnsche Bauxite Maatschappij, Paramaribo, Suriname, South America. Both companies are subsidiaries of Aluminum Co. of America.

Frank B. McKown has been appointed vice president and **Charles N. Whitaker** assistant sales manager of Kennecott Sales Corp.

Lowell B. Moon, who joined Kennecott Copper Corp.'s exploration staff in January, has been appointed district geologist for the northwestern states. **Donald D. Smythe,** formerly with the exploration dept. of Kennecott Copper Corp., has been appointed district geologist for the company's southwestern states. **C. H. Burgess** joined the exploration staff of Kennecott. He will serve as district geologist for the central states.

N. E. Nilsen is visiting the United States to investigate mining and milling practice for telluride and tungsten ores.

Alan Probert has just returned from a five week professional trip to Brazil, Peru, and Colombia in relation to Bureau of Mines participation in the Point IV program.

Robert W. Van Evera, who for the past year has been chief engineer for Reynolds Jamaica Mines Ltd., developing bauxite deposits, returned to United States with his family. His address for the present will be Crosby, Minn.

Claiborne C. Van Zandt, chief engineer of the Lone Star Cement Corp., has been made vice president of the company.

James Westfield, long engaged in safety work for the Bureau of Mines, has been named director of the Bureau's Health and Safety Division, taking over the position formerly held by bureau director, **J. J. Forbes.**

Richard V. Wyman has resigned as geologist with Cerro de Pasco Corp. and is employed by New Jersey Zinc Co. in Prescott, Ariz.

Howard I. Young, deputy administrator of the DMPA, was awarded an honorary degree of Doctor of Laws at Lindenwood College, St. Charles, Mo. Young was a director of the Colorado Mining Assn., president of the American Zinc Institute, and has, for many years, been president of the American Mining Congress.

Obituaries

Ross B. Rathbun (Member 1916), retired engineer, died on Apr. 3, 1952. He had served with the American Smelting & Refining Co. for 30 years. In 1904 Mr. Rathbun was associated with the Pacific Gas & Electric Co. For several years he was employed by various mining companies and in 1923 joined AS&R. He served the company's mines through Mexico with headquarters in El Paso. At the time of his death Mr. Rathbun was residing at Berkeley, Calif.

A. H. P. Wynne (Member 1895), Life Member, has died, according to Institute records. Mr. Wynne was born at Essex, England in 1869. He came to the United States and in 1880 was assayer for the El Trigo property in Mexico. He later became associated with the Anglo-American Mining Co. He was also connected with the Calabacillas Mining Co. and the Sierra Mining Co. At the time of his death he was residing in Mazatlan, Mexico.

Samuel Edwin Guthrey (Member) died May 13, 1952. Mr. Guthrey was born in Joplin, Mo. and attended Colorado State College from 1902 to 1906. He was assistant mining engineer with Union Basin Mining Co. at Golconda, Ariz., from 1916 to 1918. Mr. Guthrey was assistant mining engineer with Tonopah Belmont Mining Co. at Telluride, Colo. for two years. He was employed as assistant mining engineer, with Sunnyside Mining Co. at Eureka, Colo. From 1921 to 1923 he was engineer with American Carrara Marble Co., at Carrara, Nev. His last position was mining engineer with U. S. Dept. of the Interior where he did examination and reports on mining properties in Western United States and Alaska.

Thomas M. Harris (Member 1951) has died. A graduate of Carnegie Institute of Technology (B.S.) and Indiana Law School (L.L.B.), he joined the American Telephone & Telegraph Co. at Indianapolis in 1926. For one year he was a surveyor and draftsman and in 1928 joined the Link Belt Co. as a draftsman. He became sales engineer and engineer, successively.

Robert L. Hallett (Member 1916) died on May 16, 1952. He was born in Estes Park, Colo. and was graduated in 1905 from the Colorado School of Mines. Mr. Hallett became a chemist and engineer for the National Lead Co. in 1911 and was chief chemist of the company from 1938 to 1948. Since 1949 he had been a consulting mining and industrial engineer with an office at 132 Nassau St. In 1939 he became chairman of the subcommittee on tin of the Army and Navy Munitions Board. In 1940 and 1941 he was adviser on tin to the Council of National Defense. He was a member of the American Chemical Society and the American Society for Testing Materials. Mr. Hallett was a member of the Mining Club of New York and had served as its president. Surviving are his wife, Mrs. Ann Elizabeth Beyer Hallett; a daughter, Mrs. Frances Denton; two brothers, William Hallett and Alfred Hallett; a sister, Mrs. May Banford, and a grandson.

WILLIAM H. HOOVER

William H. Hoover, 63, (Member 1950), president of Anaconda Copper Co., died in June, 1952, after a long illness. President of Anaconda since 1949, Hoover once held the position of vice-president and general counsel in the company. He had been a director since 1947. Born in Lodi, Ohio, Hoover earned an A.B. degree at Wooster University in 1909, and later an LL.B. from Harvard. He joined Anaconda in 1914. President of the First National Bank of Great Falls, Mont., Hoover served as president of the City's Chamber of Commerce. He belonged to the American, and Montana Bar Associations and the National Association of Manufacturers. At one time he was district governor of Kiwanis International. He is survived by his wife, Grace Young Hoover, a son, Jack, of Belt, Mont., a sister, Mrs. Paul Morrison, of Clearwater, Fla., and a brother, Admiral Howard Hoover, of Washington, D. C.

William Green (Member 1939) died on Mar. 13, 1952. Mr. Green was with the ore sales dept. of the Cleveland-Cliffs Iron Co., Cleveland. He was born at Cleveland in 1889 and attended Yale University and Sheffield Scientific School. In 1910 he received the degree of Ph.D. and in 1911 joined Cleveland-Cliffs.

Joseph H. Rodgers (Member 1909), one of Colorado's leading mining geologists, died in Boulder, Colo. on Wednesday, the 23rd of April, 1952. Mr. Rodgers, with his brother, Myron Rodgers, was responsible for the early development of the Hidden Creek mine, at Anyox, British Columbia, and the Nickel Plate mine, also in British Columbia. He had been active in mining in Colorado since 1922, and in Boulder County for the last fifteen years. He was one of the owners of the Slide mine at Gold Hill, and while that property was being worked, it made profitable returns through his management. Recently he had been interested in a tungsten lease in the Sugarloaf district of Boulder County. Born in Charleroi, Pa., Dec. 3, 1897, Mr. Rodgers later secured degrees from both the University of Wisconsin and Northwestern University. He was president of the Northwestern Alumni Association of this region in 1941 to 1942. J. H. Rodgers was the grand old man of Colorado mining, and particularly of the Boulder County mining districts. He had the experience and ability to read the rocks, as most of us read a book, simply, directly, and without effort. With a keen interest in any new development in the industry, he was always ready to help and advise on any problem of mining or geology, and generously gave his time and energy to all who asked it.

NECROLOGY

Date Elected	Name	Date of Death
1942	Leonard Aldridge	June 8, 1952
1949	Virgil D. Basinger	Unknown
1946	W. R. Benedict	July 9, 1952
1910	Andrew B. Crichton	July 9, 1952
1949	Vernon J. Nelson	May 31, 1952
1947	John C. Reed	June 14, 1952
1919	Hugh M. Shepard	May 30, 1952

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on July 31, 1952 was 18,027; in addition 1,546 Student Associates were enrolled.

ADMISSIONS COMMITTEE

T. D. Jones, Chairman; Thomas G. Moore, Vice-Chairman; H. S. Bell, F. W. Hanson, R. H. Chadwick, T. W. Nelson, C. A. R. Lambly, John T. Sherman, A. C. Brinker, G. P. Lutjen, Ivan Given, E. A. Prentis, C. Leslie Rice, Jr., and J. H. Scaff.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Bessemer—Reed, Eugene P. (M) (R, M)
Birmingham—Breth, James E. (M)
Birmingham—Cockrell, Robert C. (J)

Arkansas
Fort Smith—McLane, Fred (A)

California
Eureka—Atkinson, James E. (M) (R. M)
Redding—Luty-Lutenko, Alexander (M)
Montebello—Lerner, Earl A. (M)
San Francisco—Barton, Thomas V., Jr. (J) (R. C/S—S-J)
San Leandro—Bares, Leonard (M)

Colorado
Pueblo—Sobral, Daniel M. (M) (R. M)

Connecticut
Cannondale—Wilson, Carroll L. (M)

Florida
Nichols—Phillips, Charles S. (J) (R. C/S—S-J)

Idaho
Kellogg—McBroom, Horace R. (J) (R. C/S—S-J)
Pocatello—Miller, James G. (M)

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Duluth—Palasvirta, Ossi E. (J) (R. C/S—S-J)
Minneapolis—Lentz, Thomas H. (J)

Missouri
Joplin—Nixon, Virgil H. (M)

Montana
Butte—Kamener, Basil A. (J) (C/S—S-J)
Butte—Palmer, William A. (J) (C/S—S-J)
Garrison—Burnet, Frederick E. (M)

Nevada
Pioche—Wigglesworth, Robert M. (M)
Ruth—Hickman, Roy F. (M)
Ruth—Seerley, John J. (M) (C/S—J-M)
Ruth—Young, Robert K. (J)

Oregon
Riddle—Foster, Walter A. (J)

Pennsylvania
Easton—Drinkhouse, David F. (J) (C/S—S-J)
Pittsburgh—Strod, Arvid J. (M)

Tennessee
Columbia—Hall, Jeptha R. (J)

Utah
Tooele—Strite, Russell R. (J)

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Austinville—Frankenfield, William D. (J) (C/S—S-J)

West Virginia
Coalwood—Waine, Charles R. (J) (C/S—S-J)
Princeton—Tunsberg, Douglas L. (M)

Argentina
Jujuy—Syons, Wilfred A. (J)

Bolivia
La Paz—Sarmiento, Marco E. (J) (R. C/S—S-J)

Canada
Asbestos, Que.—Eby, Joel O. (M)
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Ceylon
Colombo—Kularatnam, Karthigesapillai (M)

Cuba
Bayamo—Sherman, Alfred L. (M)

French Morocco
Oued—Zem-Husen, Jan W. R. (M)

Malaya
Kuala Lumpur—Richards, Owen (A)

Mexico
Estacion de Wadley, S.L.P.—Deveze, Ulises G. (M)
Federal District—Loretta, Frank B. (M)
Federal District—Perez Martinez, Jose J. (M)
Sonora—Tye, William H. (M)

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Kaikobe—Laing, Alexander C. M. (J) (R. C/S—S-J)

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Coming Events

Sept. 3-6, AIME, *Fall Meeting*, Industrial Minerals, Minerals Beneficiation, Mining, Geology, & Geophysics, Iron and Steel, and Coal Div., Palmer House, Chicago.

Sept. 3-13, AIME, Chicago Section, Centennial of Engineering, Chicago.

Sept. 5-6, Engineers' Council for Professional Development, annual meeting, Hotel Sheraton, Chicago.

Sept. 11-13, American Institute of Chemical Engineers, Palmer House, Chicago, Ill.

Sept. 13, Columbia Section, AIME, Canadian Institute of Mining and Metallurgy, joint all-day meeting, Metaline Falls, Wash.

Sept. 18, AIME, Utah Section , Newhouse Hotel, Salt Lake City.

Sept. 20, Colorado Section, AIME, Denver. Andrew Fletcher, speaker.

Sept. 22-25, American Mining Congress, Metal and Nonmetallic Mining Convention and Exposition, public auditorium, Denver.

Sept. 23-25, Institution of Mining and Metallurgy, symposium on mineral dressing, Royal School of Mines, London.

Sept. 30, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Sept. 30-Oct. 3, Iron and Steel Exposition, Cleveland Public Auditorium.

Oct. 3, AIME, National Open Hearth, Southern Ohio Section, Deshler-Wallick Hotel, Columbus, Ohio.

Oct. 10, AIME, Eastern Section, National Open Hearth Steel Committee, Harwich Hotel, Philadelphia.

Oct. 20-22, AIME, Institute of Metals Div., fall meeting with National Metal Congress, Hotel Adelphia, Philadelphia.

Oct. 24, Illinois Mining Institute, annual meeting, Hotel Abraham Lincoln, Springfield, Ill.

Oct. 28, Assn. of Consulting Chemists and Chemical Engineers, Inc., annual symposium, Hotel Belmont Plaza, New York.

Oct. 30-31, AIME, Fuels Conference, Coal Div.; ASME, Fuels Div., Bellevue-Stratford, Philadelphia.

Nov. 4, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Nov. 6-8, New Mexico Mining Assn. and **International Mining Days**, joint convention, Alvarado Hotel, El Paso.

Nov. 18, AIME, Buffalo Section, National Open Hearth Steel Committee, Hotel Statler, Buffalo.

Nov. 19, American Mining Congress Coal Div. Conference, Wm. Penn Hotel, Pittsburgh.

Nov. 20-21, American Society for Quality Control, mid-west conference, Claypool Hotel, Indianapolis.

Dec. 2, American Mining Congress, annual membership meeting, University Club, New York.

Dec. 4-6, AIME, Electric Steel Furnace Conference, Hotel William Penn, Pittsburgh.

Dec. 7-10, American Institute of Chemical Engineers, annual meeting, Hotels Cleveland and Carter, Cleveland.

Dec. 8, AIME, Arizona Section, all-day meeting, Tucson.

Feb. 16-19, 1953, AIME, annual meeting, Statler Hotel, Los Angeles.

Mar. 16-20, National Assn. of Corrosion Engineers, annual conference and exhibition, Hotel Sherman, Chicago.

Apr. 12-May 23, Empire Mining and Metallurgical Congress, Australia-New Zealand.

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